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A Pertinent Question as to Membership in the International Congress of Applied Chemistry.

The Eighth International Congress of Applied Chemistry is now only a little over a month off and the question whether to join the Congress or not becomes a burning question.

We have occasionally heard chemists ask themselves: Will it be worth while for me to join the Congress as a member? To those who will be fortunate enough to be able to attend in person, the brilliant program which has been arranged, as given elsewhere in this issue, must be convincing argument. Those members who cannot attend themselves will get at least the twenty-four volumes of the printed Proceedings—a veritable library on the present status of applied chemistry in all its phases, the cost of printing alone to the Congress being greater per member than the membership fee.

However, one point should not be overlooked. There is always a psychological moment when it becomes absolutely necessary to climb on the band wagon if one desires not to be left behind. The majority of the papers will be printed in advance and these are now in the hands of the printer. As a matter of course, the Congress had to fix the number of copies to be printed, and delivery of the printed Proceedings cannot be made to more members than the number of sets now in press. Who comes first is served first. Therefore as it is worth while to join the Congress, it is decidedly worth while to do so *now*. Mr. Wm. J. Matheson, 182 Front St., N. Y. City, is the treasurer of the Congress and the membership fee is five dollars, which is a purely nominal fee.

But, after all, the question as to worthwhileness reveals a rather narrow viewpoint. This is the first in the series of International Congresses of Applied Chemistry to be held in America. The invitation to come to America was made in the name of the "American chemists." Right at the start, the president of the Congress, Dr. Wm. H. Nichols, emphasized: "Every chemist in the United States must feel that he is part of the Congress from the very creation of the organizing committee down through all details of management to the very close of the Congress." And this principle of broadest democracy has been strictly and successfully adhered to.

The question as to worthwhileness is of decidedly secondary importance. The only pertinent question which every American chemist must ask himself is this: Can I afford not to join the Congress and thereby declare myself outside of the united democracy of American chemists?

The Value of Refined Analysis in Ore-Testing

In testing an ore to determine its amenability to a metallurgical process, it is customary to apply certain standard and well known tests. The number and nature of these tests may vary from time to time as experience and knowledge are accumulated, bût the general tendency is toward standardization. An ore is subjected to certain routine treatment, and if the re-

sults are good the process in view is adopted; if not, it may be rejected on insufficient evidence.

The policy of relying on standardized methods may be carried too far. Standardization is all right if the operator is not unduly impressed by it as an expression of finality. A rule is sometimes valuable on account of the exceptions it affords, and these exceptions should be carefully noted and investigated. Ore-testing is a work which should be characterized by individuality. Rough and ready methods of analysis will not give a clue to deep-seated troubles. Such methods may serve to control a process which is running smoothly, but refined analysis is called for to determine small quantities of certain elements and ascertain their form of combination.

A case in point is cited by A. F. Crosse, of South Africa, in discussing the effect of sulphates and arsenates on cyanide solutions. It is well known, as he says, that magnesium sulphate will decompose cyanide solutions, and that the mineral kieserite, hydrated sulphate of magnesium, is not uncommon. An ordinary analysis of an ore containing this mineral in small quantities, showing percentages of silica, iron, lime, magnesia, sulphur, etc., would be of no value to the manager of a cyanide mill. It would give no direct clue to the presence of magnesium sulphate, especially if the ore was pyritic and the sulphur was assumed to exist as sulphide of iron.

Other instances are known in which small quantities of elements, even tenths and hundredths of 1 per cent, have proved detrimental to the chemical treatment of an ore, and have had to be removed or rendered innocuous. Arsenic, selenium, tellurium, fluorine and other elements are not uncommon constituents of ores. The combinations in which they occur are relatively rare, and their presence may not be indicated without resort to refined analysis. All of which leads to the belief that a complete analysis of an ore is one of the first steps to be taken in testing it for chemical treatment. With full knowledge of original conditions, the difficulties of treatment can be anticipated and usually overcome.

Tendencies in Zinc Metallurgy

Current literature and the records of the patent office show that suggested improvements in the metallurgy of zinc are almost wholly along two lines, namely, utilizing lower grade ores, and smelting with electric heat. The former appears likely to be realized before the latter, at least in this country, because the pre-requisite of igneous concentration in the first instance is more certain and easy of accomplishment than the development of cheap electric power which is required in the second case.

One of the requirements in present practice in zinc smelting is ore of comparatively high metal content. For years the standard on sulphide concentrates produced in the Rocky Mountain region has been 40 per cent metallic zinc. In the Joplin district, the standard grade is 60 per cent. Carbonate and silicate ores are marketable if they contain 35 per cent metal, and a few instances have been known in which carbonate ores containing 20 per cent zinc have been sold at Leadville.

It will be apparent to those familiar with the conditions, that the above requirements exclude the possibility of mining and treating much ore that is not up to the standard; and further, that the physical condition of some of these ores is such that mechanical concentration is out of the question. Igneous concentration suggests itself, because the production of zinc pigment already constitutes a successful industry of some magnitude. While it is true that the production of pure pigment entails certain refinements which add to the cost of the process, these need not enter into the igneous concentration of low-grade ores for the subsequent production of spelter. In the latter case, the concentrated product may be of inferior grade as a pigment, but still of higher grade than the bulk of zinc concentrate now smelted. A charge of such oxide and powdered coal would consist mainly of volatile substances, and leave little solid residue to be disposed of.

Elsewhere in this issue are recorded two interesting proposals for improvements in zinc reduction. One is embodied in an article by F. L. Clerc, of Colorado, and the other in a patent granted to C. V. Thierry, of France. The names of both of these men are known in connection with the metallurgy of zinc. It is interesting to note the similarity in the claims which each makes for the advantages to be gained by smelting zinc oxide produced by igneous concentration, as compared with ordinary roasted concentrates containing more or less of the original gangue and some metals other than zinc. Each has devised a furnace specially adapted to the reduction of this product. A novel feature of Mr. Clerc's furnace is the method of conducting heat to the charge through a bath of molten copper which, in turn, is heated by burning suitable fuel. Mr. Thierry proposes to use an electric resistance furnace to smelt the charge.

Igneous concentration as a preliminary step in treating low-grade zinc-bearing material is not a new and untried scheme. The Pape process is in successful operation in Silesia, treating zinky slag. A method of similar intent has been tried in Colorado, but without commercial success. The slag experimented with was much lower in grade than that treated in Germany. The impression prevails, however, that there is a field for the application of this method of utilizing low-grade carbonate ores and other substances containing zinc in comparatively small proportions. Certainly it presents some attractive features in the way of securing a high-grade smelting material, and in the elimination of much of the time and labor now required to dispose of retort-residue.

Finished Steel Output in 1911

A remarkable showing is made by the partial returns of finished steel production in 1911, which have just been made public by the American Iron and Steel Association. The production of wire rods, 2,450,453 gross tons, proves to have been the largest on record, whereas the output of structural shapes, 1,912,367 tons, was the smallest since 1905, with the single exception of the year 1908, when all tonnages were very small.

It is due to the fact that wire, almost the sole product of wire rods, is ubiquitous, that the output of rods exceeded the output of structural shapes in 1911 by so large a margin, 28 per cent. The year was not one of large projects. The railroads were very conservative in their expenditures, and there was not very active erection of skyscrapers and other large buildings. Normally, of course, the railroads are large consumers of structural steel, while they are not conspicuously

large consumers of wire. It has always been observed that in a "bad year" the tonnage of wire stands out conspicuously. This was the case in 1908, when the total of rolled iron and steel declined from 1907 by 40 per cent. and the wire rod output declined only 10 per cent. In 1911 production of pig iron decreased 13 per cent production of steel ingots and castings 9 per cent and the output of wire rods increased 10 per cent. These are comparisons of 1911 with 1910, but it should be observed that while in 1910 new records were made by pig iron and steel, the rod output had been greater in 1909 than in 1910, so that the rod output of 1911 exceeded the record output by only 6 per cent.

Conditions in the present year present a reversal of those of 1911. The demand for fitted structural work and for railroad cars has been exceptionally large, while, relatively speaking, the wire trade has languished. It may show a larger tonnage this year than last, but structural shapes are certain to show a very large increase.

The Condensation Problem in Electric Zinc Smelting.

In this issue we publish an able and interesting Canadian Mining Institute paper by Mr. Walter Renton Ingalls on the electric smelting of zinc ore. There is no need to say anything of Mr. Ingalls' standing as one of the foremost zinc metallurgists of the present time. On this account the stand which this authoritative and conservative representative of the old classical school of zinc metallurgy takes with reference to electric-furnace insurgency is decidedly interesting, so that his paper should prove interesting reading.

His statement in last year's *Mineral Industry* that "the electric smelting of zinc ores is not only still in the experimental stage, but is in the infancy of the experimental stage," is not repeated now by Mr. Ingalls. His paper is clear evidence of the great pains which he has taken to be fair and impartial. For this reason we will not enter into any controversy with respect to quite a number of the author's statements, with which we do not agree at all. Mr. Ingalls is "not yet by any means prepared to say that electric smelting will offer any advantage over existing methods, even if difficulties that are now in view can be overcome." We have taken repeatedly and decidedly the opposite stand and have endeavored to prove the advantages that must result from the large continuously working electric zinc furnace when all the present difficulties have been overcome.

There is no need of repeating our argument at this place now; the evolution of electric zinc smelting has gone on and is proceeding along logical lines, and nothing has happened that would induce us to modify our views. Qui vivra verra. For the present even insurgents must concede that standpatters have a right to live—*and vice versa*.

The object of this note is to discuss one single phase of electric zinc smelting on which Mr. Ingalls lays great stress, and deservedly so. This is the difficulty which almost all investigators of electric zinc smelting have more or less encountered in condensation. Instead of liquid zinc blue powder is largely produced. Mr. Ingalls limits himself to the mere statement of the fact and purposely refrains from advancing hypotheses as to the reason. It is quite true that this is quite a complicated matter, and from case to case the reasons for the formation of blue powder may differ. But it is quite possible to state concisely without any hypothesis certain conditions which must produce blue powder.

We think it was Mr. W. McA. Johnson, the electric zinc furnace pioneer, who distinguished between "physical" and "chemical" blue powder. What these catch words mean is clear. Blue powder may be produced by special purely physical conditions or by special purely chemical conditions. This is a really useful distinction.

As to "physical" blue powder, the analogy with snow is obvious. Just in the same way in which water vapor may condense into snow instead of rain, zinc vapor may condense into blue powder instead of liquid zinc under special conditions of pressure and temperature. This is simply a problem of physical chemistry and it would seem to present opportunities for more than one thesis with experiments on vapor pressure of zinc and the conditions of equilibrium between liquid and gaseous zinc or solid and gaseous zinc. But when the physical chemistry of the problem is completely understood, the chemical engineer, with a mechanical engineering instinct, will still have some interesting work to do in the design of the condenser.

With respect to the formation of "chemical" blue powder, the primary supposition is that from the vapor not merely zinc but something else is condensed. As mentioned in Mr. Ingalls' paper, Boudouard found that when a certain limit of carbon dioxide in the gas of the zinc retort is exceeded carbon dioxide acts oxidizing upon the zinc, and coating the condensing globules of zinc prevents coalescence and forms powder. Now, many impurities in the ore under treatment will act in the same way. For instance, if the ore contains silica and an arc furnace is employed, silica or silicon vapors will be formed, and in condensation each condensing globule of zinc will be coated with a fine layer of silica and the coated globules will not run together and form liquid zinc, just in the same way as it is impossible to run globules of mercury coated with flour together into a liquid mass of mercury.

This would explain why Mr. Weeks, in his Schenectady experiments had no difficulty at all in condensation, as he treated zinc dross and thus had a rather pure vapor. It would also indicate an advantage of the resistance furnace in permitting to keep the temperature low enough to prevent pernicious impurities from passing into the vapor. In connection with arc furnaces it would explain the necessity of passing the vapor through some reducing agent like a column of incandescent carbon, as used by Mr. Johnson, in order to eliminate as much as possible all impurities which might interfere later on in the condensation of the zinc.

Thus it will be seen that the formation of blue powder necessitates the maintenance of correct physical and chemical conditions, and it is clear that the difficulties of the problem are enhanced by any increase of furnace unit and condenser unit. In any case where condensation difficulties are experienced it will be of fundamental importance to establish the reasons why and especially to ascertain whether the blue powder formation is due to faulty physical or chemical conditions. Though the problem is complicated, there is no reason why it should not be solved as a matter of course and in due time.

The Iron and Steel Market.

July in the steel market furnished a surprise. While sellers were prepared for a decided falling off in business, specifications have proved practically as heavy as in June. In view of the fact that July and August are almost invariably dull months, and the further fact that many low priced contracts expired with June, a decided decrease in specifications was to be expected. There is reason now to suspect that buyers are anticipating their requirements, being moved to do so by the fact that mills are falling behind in deliveries. The average specification for steel bars now requires about six weeks to fill, and deliveries of universal mill plates are almost as far behind. Such anticipation of requirements, while perhaps necessary for buyers when mills fall behind, tends to create an unsound position, for it accentuates the conditions which the buyer wishes to protect himself against, and the movement sometimes grows like a snowball, the experience in the second half of 1909 being a good illustration. While the market is strong, there are suggestions now being made in some quarters that there is danger of a runaway market.

There are predictions that bars, plates and shapes will shortly advance a dollar a ton, following the similar advance of June 26, which was to 1.25 cents for bars and 1.30 cents for plates and shapes.

Under date of July 20 the American Steel & Wire Company advanced its products $1 a ton, making plain wire 1.45 cents, galvanized wire 1.75 cents, wire nails $1.65, painted barb wire 1.65 cents and galvanized barb wire 1.95 cents. The advance was promply followed by the independents. Buyers are well covered by old contracts, some of them made before the previous advance.

Pig Iron.

There has been a general advancing tendency in pig iron in all markets, although sales have not been heavy. Southern iron has advanced fully 50 cents, being available at $11.50, Birmingham, only for prompt shipment, while $12 is demanded for late deliveries. In the north the average advance has been about 25 cents. We quote foundry iron at $15.50 delivered Philadelphia and at $14.50 to $15 f. o. b. Chicago district furnaces. Valley prices are: Bessemer, $14.25 to $14.50; basic, malleable and No. 2 foundry, $13.50; forge, $13 to $13.25.

Steel.

Steel has become scarcer but prices have advanced only slightly. Consumers are well covered by contracts, and are only buying as their requirements exceed the deliveries they can secure on contracts. The mills are unwilling to quote for fourth quarter delivery on billets and sheet bars, and evidently expect a materially higher market. For third quarter delivery billets are quotable at $21.50 to $22 and sheet bars at $22 to $22.50 f. o. b. maker's mill, Pittsburgh or Youngstown. Rods are $25, Pittsburgh.

Finished Steel.

Effective July 1 steel pipe larger than 6-in. was advanced one point or about $2 a ton, matching the advance of June 1 on 6-in. and smaller. Iron pipe has been advanced similarly. Regular discounts, on "jobbers' carloads," sizes ¾ to 1½-in., are 80 per cent off for steel pipe and 74 per cent off for iron pipe.

Current prices, f. o. b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Light rails, 16 and 20-lb., 1.24 cents; 25, 30, 35, 40 and 45-lb., 1.19 cents.

Plates, tank quality, ¼ in. and heavier, 1.30 cents.

Shapes, 1.25 cents.

Steel bars, 1.25 cents.

Iron bars, 1.35 cents, Pittsburgh; 1.35 cents, Chicago; 1.32½ cents, Philadelphia.

Sheets, blue annealed, 10 gage, 1.45 cents; black, 28 gage, 2.00 to 2.05 cents; galvanized, 28 gage, 3.05 to 3.15 cents; painted corrugated roofing, 28-gage, per pound, 2.15 to 2.20 cents; galvanized, 3.10 to 3.20 cents.

Tin plates, $3.50 per box for 100-lb. cokes.

The Non-Ferrous Metal Market

Since our last report the market for copper and tin has shown unusual activity, followed by dullness after the national holiday. Conditions in European markets have affected the domestic market more than usual, especially in tin and copper. Lead has been arbitrarily advanced by the leading seller, and an average business has been transacted.

Copper.—Late in June copper sold at 17⅝ cents, but early in July the demoralization of the London Market caused a decline in the European price to 16½ cents, with the result that no foreign business could be transacted by sellers on this side, and the domestic market weakened. At last reports Lake copper has a nominal quotation of 17⅛ to 17⅜ cents. Electrolytic is quoted at 17 to 17.05 cents.

Tin.—The London dock strike brought about an acute situation in the New York tin market. Spot tin commanded a premium of 3 cents until July 9, when the premium was reduced. The close for July tin at last reports was 44⅜ cents, New York.

Lead.—On July 3 the price was advanced from 4.50 to 4.75 cents, New York. St. Louis advanced to 4.62½ to 4.65 cents. The market has been firm and good business is reported.

Spelter.—Consumption was excellent early in July, but later the principal buyers staid out of the market, and business was dull. Some premiums were realized on July spelter, as the supply was short, but little or no business was reported in September spelter. The last quotations are 6.95 to 7.05 cents, St. Louis, and 7.10 to 7.20 cents, New York.

Other Metals.—Consumption in aluminium continues good and the quotation is firm at 22½ to 23 cents, New York. Larger sales of antimony are reported and prices are firmer, ranging from 7.25 to 8.50 cents for various brands. Quicksilver has been quiet and there is little change in prices. New York quotations are $42.50 per flask of 75 lb. San Francisco reports quotations of $42.

The Western Metallurgical Field

Teaching Metallurgy at Missouri.

At the Missouri School of Mines and Metallurgy, assaying, metallurgy and ore dressing are taught in the same department. In designing the laboratories, the teaching staff kept constantly in mind the idea of teaching the principles of the subjects involved. Adaptability for commercial testing or for much full-scale experimenting has been considered of secondary importance. The result is believed to embody excellent testing laboratories, with some opportunity for full-scale work, but with special advantages in teaching metallurgical principles. The latter feature is believed to be absolutely essential to the successful work of graduates, and the equipment has been selected and the courses planned so that fundamental principles may be impressed on the student. Practically all the laboratory work in metallurgy is done by the student without laboratory directions before him on a printed sheet, so that he finds it necessary to post himself on the nature of the work before he undertakes it.

The assay laboratory is equipped with coal, coke and gasoline furnaces of standard design. The coal furnaces and bins are so arranged that it is easy for a student to see whether he is using fuel economically, or whether his neighbor is getting better results from less coal. The furnaces are adapted to the use of pyrometers and gas-analyzing apparatus, so that the students can make these determinations when desired. The stock room is well arranged for storing samples of ore. These are put in sacks holding about three ounces each. Original

samples weighing from 20 lb. up are carefully ground and assayed before being given to the students.

In the metallurgical laboratory there is a room for small-scale experiments. One part of the laboratory is occupied by a water-jacket furnace, 20 in. at the tuyeres and having a smelting column of 6 ft. This can be used to teach the principles of smelting. A crew of students is required to get ready the data necessary for a smelt, make the calculations, prepare the furnace, blow in, recover the products, blow out, sample and assay the products and take account of stock. By this means it is possible to reinforce the class room instruction in a way that would not be possible without the furnace. A hand-reverberatory roasting furnace and an improvised blast-roasting apparatus are provided for work of this kind. A small zinc smelting furnace is being constructed, to be used principally in emphasizing the principles of zinc smelting. A cyaniding laboratory is provided in which small-scale operations can be conducted, and the chemical principles shown.

The ore dressing laboratory contains a complete equipment for crushing, sizing and concentrating ores of all kinds. The machines in this department are full size, and are so arranged that they can be operated independently or together. In laboratory practice the student is required to work with each machine until he is familiar with the principles underlying its construction and operation. After he has become familiar with these points, he is required to use the machines in groups, and is given experiments of increasing difficulty until a complete mill scheme is carried through.

The metallography laboratory will accomodate 12 students. It is equipped with suitable miroscopes and accessories, photomicrographic camera, and other instruments needed in the microscopic study of metals. A specimen cabinet contains 1200 specimens, with a complete history of each specimen. Photographic plates also are filed and indexed.

In a bulletin recently issued from the school at Rolla, Mo., the faculty of the department of metallurgy has published a detailed statement of the work done in metallurgical instruction. Examples are given of the type of problems given the students in each branch of the work. The nature of the problem and the method of working it are designed to develop personal observation and reasoning, instead of merely telling the student what to do and what the result should be. On the whole, the article makes out a very good case for the methods used in developing young metallurgists at Missouri.

Losses by Fire.

The mine buildings of the Silver Lake mine, near Silverton, Colo., were destroyed by fire recently. The fire originated in the blacksmith shop while the employes were at dinner and spread rapidly to adjoining buildings. The boarding house was saved by dynamiting buildings between it and the fire, but the upper terminal of the long tramway which connects the mine and concentrating mill was destroyed. This resulted in stopping operations at the mill. The loss is reported to be between $30,000 and $40,000. The property is owned by the American Smelting & Refining Company, and the ore produced is smelted at the Durango smelter of that company. The fire will in no way hamper the smelter operations, as a large stock of smelting material has been accumulating at the smelter during the months following a cessation of work after the floods last spring. The company will immediately rebuild and commence operation as soon as possible.

Tonopah, Nev., was visited by a disastrous fire on July 8. The principal business section was completely destroyed, and many small buildings burned before the fire could be put out. The loss is estimated at over $300,000. Tonopah is at present a prosperous mining town. A number of companies have recently built mills there, and have adopted the latest ideas in metallurgical treatment. There is little doubt, therefore, but that the town will be rebuilt.

Revival of Interest at Marysvale, Utah.

The Bully Boy-Webster group of claims at Marysvale, Utah, includes some of the earliest locations made in that district. The mines have been worked intermittently for many years, but more recently they have been idle on account of the low grade of the ore. A new mill has been built along lines which are indicated by other successful mills on similar ore, and it is expected that the system of concentrating and cyaniding will solve the problem. A large dump will be treated first, followed by fresh ore taken from the mine in the course of development work. The ore will be screened and the larger pieces washed and hand-picked. The ore recovered from the picking belt will be crushed and added to the original fine ore. Chilean mills will be used to grind the ore to 30-mesh size, after which it will be classified and concentrated. The tailing from concentration will be cyanided by agitation, then classified into sand and slime. The sand will be discharged to the dump and the slime will be filtered on Portland filters. Zinc dust will be used for precipitation. The mill will be operated by water power, which is available practically throughout the year.

Company Reports.

The May report of the Goldfield Consolidated Mines Company was delayed in order to receive returns from ore shipped to the smelter of the International S. & R. Company, Tooele, Utah. It seems that ore carrying considerable copper and silver is being mined from the deepest workings, and is unsuitable for treatment in the mill. It is, therefore, marketed separately. The report states that the May production was 30,215 tons, containing precious metals valued at $550,656, or an average of $18.22 per ton. The mill extraction was 91.44 per cent, and the net recovery from all ore treated and shipped amounted to $16.69 per ton. Operating costs amounted to $6.60, leaving a net profit to the company of a trifle over $10 per ton. The estimated June production is 32,360 tons, being the largest monthly production in the history of the company.

The annual report of the Greene Cananea Copper Company for the calendar year 1911 has just been issued. It shows that the company treated in its mill 528,412 tons of ore produced from company mines and 113,213 tons from Cananea Duluth mines. The former had a concentration ratio of 2.4 into 1, and the latter 5.17 into 1. The operating and repair cost was $0.646 per ton, and improvements cost $0.076 per ton, giving a total cost for the year of $0.722 per ton. The average saving of copper in the concentrates was 77.69 per cent. Every pound of tailings, whether of sand or slime, is being impounded, and there is no doubt that the reclamation of clear water from the tailings pond is of more value than the cost of impounding the tailings. The concentrators are in excellent condition, the old wooden jigs having been replaced by reinforced concrete jigs. Many other small improvements have been made, but none of great magnitude.

In the reduction division the cost of smelting was $2.57 per ton. This is not quite down to the predicted mark of $2.25 per ton, but the average grade of matte has been much lower than formerly, the converting department has been congested, and the amount of by-products to be handled was greatly increased. Changes are being made in the converting department, however, by which it is still hoped to reduce the cost of smelting to $2.25 per ton of material. In the segregated costs, converting amounted to $0.6518, being a large proportion of the total cost. Blast furnace cost was $1.4055; calcining, $0.051; reverberatory, $0.3197; sampling, $0.0222; bedding, $0.0584. The cost of McDougal roasting for the year was $0.40 per dry ton.

A new reverberatory furnace was built and put in use during the year. Twelve-inch buckstays were used in place of the old 10-inch; and a 20-inch arch was built in place of the previous 12-inch arch. The cost of reverberatory smelting

amounted to $1.65 per ton of charge, as compared with $1.90 for the previous year. Without the cost of rebuilding No. 1 furnace, the cost was only $1.427 per ton of charge. Basic lining has been introduced in the converter department, but investigation has demonstrated that the barrel-type shells are not as well adapted to economical work as the Great Falls type, and two of the latter have been ordered. This will increase the capacity of the converter department, and it will be able to produce from eight to nine million pounds of copper per year. Blowing-engine capacity for the blast furnaces will be increased as the sulphur content of the newly developed ores requires more oxygen for smelting.

Copper costs for the year were as follows: Gross cost, f.o.b. Cananea, 9.568 cents per pound; freight to New York, 1.343 cents; total, 10.911 cents per pound. The value of the precious metals credited to this gross cost amounts to 1.823 cents per pound, leaving a total cost of fine copper sold of 9.088 cents per pound. The total cost of mining and treating a ton of ore, including every cost until the refined copper has been sold, was $5.257. The yield of copper per ton of ore treated was 50.01 pounds. It is probable that during the coming year the yield will not be so high.

Metallurgy as She Is Promoted.

Metallurgy is not without its humorous side. The pompous promoter who uses technical terms with reckless abandon, the ignorant tyro who flounders in a subject that is too deep for him and the crafty schemer who uses his knowledge to play on the credulity of those less well informed, all contribute to the gaiety of the profession. Occasionally these men are vicious, but more often they are sincere and exhibit a courage and assurance that is equaled only by their ignorance. In developing their processes they draw on a vivid imagination, occultism and various forms of well-known stimulants. Incidentally they may use electricity or an acid, but these are mere matters of detail and the main reliance usually is placed on something less tangible.

Some time ago one of these processes was brought to the attention of one of our large metallurgical concerns. It purported to be a process "for the disintegration of metals by vibration," whatever that may be, and promised marvelous results. No demonstration was made, but a meeting was arranged between the promoters and the company's metallurgist, at which the details of the process were unfolded. In reading the report on the process which the metallurgist made to his principals, it will be apparent that he is endowed with a strong sense of humor. The report is given below exactly as he made it, only the names being changed, of course:

"If the magnitude of the vibrations of the metals in your ores can be increased by our chemical stimulus, all of these metals will then pass into the matte and be saved. At present you solidfy; we disintegrate. These vibrations are controllable, my dear sir; that is the entire secret."

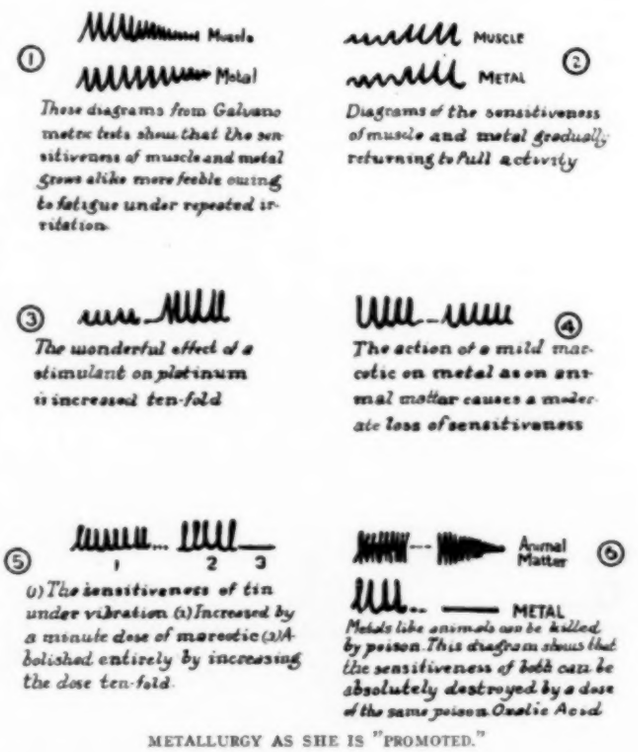
This was the dictum of Colonel Blank, Soldier of Fortune. Colonel Blank had been writing voluminous letters to Judge Breaudeigh about the recovery of platinum, iridium and other rare metals from our slag. As the colonel was very emphatic that a wet assay was needed, since his process recovers values which no fire assay would show, I judged it best to see him in person.

The colonel is a neat looking, stout man, of say, 60, with a pair of gold eye-glasses on the extreme tip of his nose. He is the man of business—"In touch with outside capital, sir." He has an assistant or accomplice named Peele, who does the actual work. Peele is a hairy, ill-kempt man, who, I judge, sleeps in his shirt. The third of the trio, the inventor, was not at the conference, he having fallen from a street car recently, probably while under the influence of the vibration.

This story goes back to the time when the inventor was a sergeant-major in India, where he met a very holy man, Prof-Jagadis Chunder Bose (not Boose, but Bose with a long o). This learned pundit has sent forth from the University of Calcutta a book entitled "Life in Metals," and to one chapter in this, called "The Response of the Living and Non-Living," your attention is particularly directed.

(At this point the waiter performs his manipulations and we place ourselves in tune with the infinite.)

The inventor sits at the feet of the aforesaid Bose and learns from him that muscle and metal are practically alike in their response to stimulants and narcotics. For example, Fig. 1 proves that muscle shows, under fatigue, a certain vibration, the amplitude of the vibration growing less and less as fatigue continues. Metals, as the diagram shows, have an almost similar action under fatigue. Fig. 2 shows that as this cause of fatigue is removed, the muscle and the metal regain their full vibrations. Passing now one step further, we learn that metals can be aroused by a stimulant and calmed by a sedative. Fig.


Muscle
Metal
These diagrams from Galvano metric tests show that the sensitiveness of muscle and metal grows alike more feeble owing to fatigue under repeated irritation.
MUSCLE
METAL
Diagrams of the sensitiveness of muscle and metal gradually returning to full activity
The wonderful effect of a stimulant on platinum is increased ten-fold
The action of a mild narcotic on metal as on animal matter causes a moderate loss of sensitiveness
1 2 3
(1) The sensitiveness of tin under vibration (2) Increased by a minute dose of narcotic (3) Abolished entirely by increasing the dose ten-fold.
Animal Matter
METAL
Metals like animals can be killed by poison. This diagram shows that the sensitiveness of both can be absolutely destroyed by a dose of the same poison Oxalic Acid

METALLURGY AS SHE IS "PROMOTED."

3 shows the wonderful effect of a stimulant on platinum. Its irritability is increased ten-fold. On the other hand, Fig. 4 shows the action of a mild narcotic on metal, causing a moderate loss in sensitiveness. Coming now to the most interesting part, we learn from Fig. 5 that any metal, tin, for example, may be stupefied by a dose of poison—oxalic acid—and that on increasing the dose the metal dies, is defunct; its vital spark is extinct, and the undertaker must be called in. So Fig. 6 closes the story with a summary showing the absolute similarity of metal to muscle in its relation to stimulant and sedative.

At this point the inventor begins to perceive a practical application of this theory to modern metallurgy, and coming to America hires Mr. Peele, who was a corporal in some volunteer regiment. Colonel Blank is admitted to the secret, as the poor old inventor has no idea of business. "In fact, my dear sir, he is a mere child in the affairs of the world." Blank becomes the business manager of the concern.

(Here the colonel drops his eye-glasses off the tip of his nose into his tumbler, which proves to be empty, and the waiter comes running with the necessary chemicals.)

Now, as to the practical side of it. "Your process tears down, we build up; you solidify, we disintegrate. We have

here a table of assays made by the great Doctor Lodeux, probably the best analyst in America, showing our results. Here is No. 12, black sand from Atlin, an ore of which we treated 6 oz., and by our process reduced it to 16 dwt. 3 gr., in which Dr. Lodeux finds a total weight of metal per ton of ore of 3918 oz. 10 dwt. 15 gr. Out of this we got 376 oz. 10 dwt. 9 gr. gold, valued at $7,530, and 224 oz. 8 dwt. platinum worth $8,554; making a total recovery of 820 oz. 18 dwt., valued at $16,084 per ton, and that, mind you, from material they were throwing away.

"Sample No. 70, to which your attention is particularly directed, is a marble, an ordinary building stone. This the process reduced from 12 oz. to a button weighing 4 dwt., containing 14 oz. platinum per ton, valued at $490, and 13 oz. 10 dwt. iridium worth $536, or a total value of $742.70 per ton of marble. Yes, sir.

"Now as to your matte. You say you only recover one ton of matte from 17 tons of your ore. We took 16 oz. of your ore and by our process of disintegration made 12 oz. of matte out of it, a recovery of 75 per cent, whereas your recovery is 17 into 100—say 6. So we give you ten times the recovery you get. Now, what is it? That's for you to say."

"Yes," says Peele, "that's right. I done it myself and I added the chemicals that disinterrogate the Silica, and I thought it must be wrong when I got such a big button, but the old man said to put it back and leave it under vibration all night and the next morning the Silica was all disinterrogated and there's your values."

"It's a flux we sell you," says the colonel. "You see you kill the ore by your process; it's dead, and that's why you lose so much of the slag. We stimulate it, invigorate it—as you can see by the diagram—its vibrations are increased tenfold; it passes into the matte. You get all of it, absolutely all. *But it must be a wet assay*, fire assays are no good. Yes, thank you. Dewar's, and another split Polly."

Program of the International Congress of Applied Chemistry.

The following is a concise program of the Eighth International Congress of Applied Chemistry, to be held in this country from September 4 to 13, 1912. Further details, especially on entertainments and excursions may be found in Announcement No. 4, issued by the Secretary of the Congress, Dr. Bernhard C. Hesse, 25 Broad Street, New York City.

Meeting at Washington.

The inaugural meeting of the Congress will be held on Wednesday, September 4th, 11 a. m., at Washington, D. C., presided over by the President of the United States.

In the afternoon there will be general excursions to the Capitol from 2 to 4 p. m., a garden party on the lawn at the White House from 4:30 to 6:30 p. m., and a reception at the New National Museum from 8 to 11 p. m.

The morning of Thursday, September 5, will be devoted to visits of the Art Galleries, the Library of Congress, the Government laboratories, or a trip to Mount Vernon. In the afternoon at 4:55 and at intervals thereafter, special trains will take the members of the Congress to New York City.

Meeting at New York City.

The main work of the Congress will be done in sectional meetings, which will be held in Columbia University.

Sectional meetings will be held on Friday, September 6, from 10 a. m. to 12 a. m. and from 1 p. m. to 3 p. m., on Saturday from 10 a. m. to 12 a. m., and during the following week on Monday, Tuesday, Wednesday, and Thursday, September 9 to 12, every day from 10 a. m. to 12 m. and from 1 p. m. to 3 p. m., at Columbia University.

The Congress will be wound up officially on Friday, September 13, by a meeting of the International Commission of Congresses of Applied Chemistry at 10 a. m. and the final General Assembly of the Congress in the great hall of the College of the City of New York at 11:30 a. m.

There are twenty-four sections and sub-sections of the Congress, as follows:

Section	I:	Analytical chemistry.
"	II:	Inorganic chemistry.
"	IIIa:	Metallurgy and mining.
"	IIIb:	Explosives.
"	IIIc:	Silicate industries.
"	IV:	Organic chemistry.
"	IVa:	Coal tar colors and dye stuffs.
"	Va:	Industry and chemistry of sugar.
"	Vb:	India rubber and other plastics.
"	Vc:	Fuels and asphalt.
"	Vd:	Fats, fatty oils, and soaps.
"	Ve:	Paints, drying oils, and varnishes.
"	VIa:	Starch, cellulose, and paper.
"	VIb:	Fermentation.
"	VII:	Agricultural chemistry.
"	VIIIa:	Hygiene.
"	VIIIb:	Pharmaceutical chemistry.
"	VIIIc:	Bromatology.
"	VIIId:	Physiological chemistry and pharmacology.
"	IX:	Photochemistry.
"	Xa:	Electrochemistry.
"	Xb:	Physical chemistry.
"	XIa:	Law and legislation affecting chemical industry.
"	XIb:	Political economy and conservation of natural resources.

General Lectures and Joint Sessions with the American Electrochemical Society and the American Institute of Mining Engineers.

The following general lectures will be delivered:

Gabriel Bertrand, of Paris, France: "Du role des infiniment petits chimiques en chimie biologique" on Friday, September 6, 4 p. m., in the great hall in the College of the City of New York.

Samuel Eyde, of Kristiania, Norway: "Oxidation of atmospheric nitrogen and development of resulting industries in Norway" in a joint session of sections II, VII, Xa, and Xb, on Saturday, September 7, 2 p. m., in the auditorium of the American Museum of Natural History.

Carl Duisberg, of Elberfeld, Germany: "Die neuesten Arbeiten and Probleme der Chemischen Industrie," on Monday, September 9, 4 p. m., in the great hall of the College of the City of New York.

William Henry Perkin, of Manchester, England: "The permanent fireproofing of cotton goods" on Tuesday, September 10, 4 p. m., in the great hall of the College of the City of New York.

H. A. Bernthsen, of Ludwigshafen on the Rhine, Germany: "Synthetic Ammonia," in a joint meeting of sections II, VII, Xa, and Xb, Wednesday, September 11, 10:30 a. m., at the College of the City of New York.

Giacomo Ciamician, of Bologna, Italy: "La Foto Chimica dell' Avvenire," Wednesday, September 11, 4 p. m., in the great hall, the College of the City of New York.

Ira Remsen, of Baltimore, Md.: "Priestley in America," Thursday, September 12, 4 p. m., at the College of the City of New York.

The afternoon lectures at the College of the City of New York on Friday, Monday, Tuesday, Wednesday and Thursday will be preceded by an organ recital at 3:15 and followed by an informal tea at 5 o'clock.

Joint meetings will be held by Section IIIa of the Congress with the *American Institute of Mining Engineers* and by Section Xa with the *American Electrochemical Society*.

A joint meeting of Sections IIIa and Xa of the Congress, the

American Electrochemical Society and the American Institute of Mining Engineers will be held on Saturday, September 7, at 10 a. m. and will be devoted to the reading and discussion of papers of electrometallurgical interest. Among the authors who will address this meeting are: Mr. Lawrence Addicks, Dr. Richard Amberg, Mr. F. A. J. FitzGerald, Mr. A. E. Greene, Mr. Joseph O. Handy, Dr. Paul Héroult, Mr. F. T. Snyder, and Mr. C. H. vom Baur.

Joint meetings of Sections II, Xa, and Xb of the Congress with the American Electrochemical Society will be held on Monday, September 9, in the forenoon and afternoon. The feature of the afternoon session will be an experimental lecture by Dr. E. Weintraub, of the General Electric Company, on the *element boron*, its compounds, and uses.

A joint meeting of Sections VIa and Xa with the American Electrochemical Society on the forenoon of Tuesday, September 10, will be devoted to a symposium of papers on the *electrolysis of sodium chloride* for the production of caustic soda and chlorine, and hypochlorites. Among the authors who will address this meeting are Mr. A. A. Claflin, Mr. R. E. Gegenheimer, Prof. Ph. A. Guye, Dr. R. Taussig, Dr. Wm. H. Walker, Mr. Jasper Whiting and Mr. E. Wilderman.

For the afternoon of Tuesday, September 10, another joint session of Section Xa and the American Electrochemical Society is scheduled.

On Wednesday, September 11, in the forenoon and afternoon joint sessions of Sections Ve and Xa will be held, devoted to a symposium on *corrosion and its prevention*. Papers on this subject will be presented by Mr. James Aston, Prof. Chas. F. Burgess, Dr. J. K. Clement, Mr. H. H. Hanson, Mr. J. H. Lawrie, Mr. K. Lewis, Dr. E. Liebreich, Mr. W. B. Schulte, and Mr. Maximilian Toch who will present an illustrated lecture on the condition of the battleship Maine after 13 years' submersion.

A great many more joint meetings between various other sections have been arranged, but the program of the papers has not yet been finally arranged. Further details must be reserved for our next issue.

Entertainments.

In advance of the official beginning of the Congress the following informal entertainments will be held.

A reception of the Society of Chemical Industry will be held on *Saturday*, August 31, at 8:30 p. m., at the Chemists' Club. On the same evening members of the Verein Deutscher Chemiker will gather informally on the roof garden of the Hotel Astor.

For *Sunday*, September 1, the Verein Deutscher Chemiker has arranged an automobile trip to Coney Island and in the evening a concert on the roof of the New Century Theater.

On *Monday*, September 2, 10 a. m., the Verein Deutscher Chemiker will hold a business and professional meeting at Havemeyer Hall, Columbia University, where Dr. F. Raschig will speak on the scientific and technical importance of tarphenols and Dr. F. W. Frerichs on 32 years in the United States as a chemist and manufacturer. For the evening the Verein has arranged a Kommers and for the ladies a theater party.

On the same evening the Chemists' Club will give a reception at its Club House to members of the Congress speaking the Latin tongues.

On *Tuesday*, September 3, the Society of Chemical Industry will hold its business and annual meeting at the Chemists' Club. An address of welcome by the Mayor of the City of New York and the address of the retiring president, Dr. Rudolph Messel, are scheduled for this occasion.

Wednesday and Thursday are the Washington days, as noted above.

On the evening of *Thursday*, September 5, the American Institute of Chemical Engineers will hold a banquet at the Chemists' Club.

The program of entertainments during the New York days of the Congress is as follows:

Friday, September 6, 8 p. m.: Kommers to Section VIIIb by the Apotheker Verein, of New York City, 9 p. m.: Reception to Japanese delegates by Dr. and Mrs. Takamine.

Saturday, September 7, 8:30 p. m.: Reception at the Metropolitan Museum of Art in Central Park, given by the American Chemical Society to the members of the Congress.

Sunday, September 8, 10 a. m.: Boat excursion up the Hudson to Newburgh.

Monday, September 9, 7 p. m.: Garden party at Bronx Park.

Tuesday, September 10, 8:30 p. m.: Reception by the Chemists' Club to all members of the Congress at the Club House.

Wednesday, September 11, 8 p. m.; Entertainment at the Hippodrome.

Thursday, September 11, 7 p. m.: Grand banquet at the Waldorf Astoria.

Friday, September 13, 3 p. m.: Baseball game, New York vs. St. Louis at Polo Grounds. 7 p. m.: Subscription dinner of Society of Chemical industry at the Waldorf-Astoria.

For *Saturday*, September 14, visits to the Consolidated Gas Company and the Nichols Copper Company have been arranged for the forenoon, and a visit to the Corn Products Company, the Standard Oil Company, and the Bush Terminal Company for the afternoon.

On *Monday* morning, September 16, excursion trains will carry those members who participate in the two arranged trips out of New York City. The "short trip" (10 days) includes Philadelphia, Pittsburgh, Niagara Falls, Detroit, Chicago, Cleveland, and Boston. The "long trip" (37 days) includes Philadelphia, Pittsburgh, Niagara Falls, Detroit, Chicago, Omaha, Denver, Salt Lake City, San Francisco, Los Angeles, Grand Canyon, El Paso, Fort Worth, Sulphur, La., New Orleans, Birmingham, Atlanta, Ducktown, Tenn., Asheville, N. C., Charlotte, Great Falls, S. C., Danville, Va., and Washington, D. C.

For the *entertainment of ladies* during the Congress a very elaborate program has been arranged. For details see announcement IV, issued by the Secretary.

The membership fee is $5. *Nobody who intends to join the Congress should delay doing so any longer, because delivery of the printed Proceedings of the Congress (which will comprise several volumes and will indeed form in itself a veritable library on the present status of applied chemistry worth a great deal more than $5.) can be guaranteed only to those members of the Congress who join before the issue already contracted for is exhausted.* (All the papers which were received by July 1 and accepted are now in the hands of the printer.) The membership fee of $5 is to be sent to Mr. *W. J. Matheson*, Treasurer, International Congress of Applied Chemistry, 182 Front Street, New York City.

Joint Resolution for Patent Office Investigation.

Representative Bulkley, of Ohio, on July 15th introduced in the House of Representatives a joint resolution requesting the President to order an investigation of the Patent Office, "to investigate fully and carefully the administration of the Patent Office with a view of determining whether or not the present methods, personnel, equipment and building of said office are adequate for the performance of its functions, taking into consideration the present character and volume of business, and also such increase in complexity or volume as may reasonably be expected in the future, and to ascertain and recommend specifically to Congress not later than Dec. 10, 1912, what changes in law, what increases in appropriations and what additional building accommodations may be necessary to enable the Patent Office to discharge its functions in a thoroughly efficient and economical manner, and to what extent any expenditures which may be recommended can be met by increases of Patent Office fees."

A Microscopic Study of California Electric Furnace Pig Iron

BY D. A. LYON AND F. C. LANGENBERG.

A sample of pig iron was recently received from the Noble Electric Steel Company, which was made in an electric furnace at their plant at Heroult, Shasta County, Cal. The ore used in the production of this iron shows the following analysis:

Fe	69.9	per cent.
Fe_3O_4	89.4	per cent.
Fe_2O_3	7.3	per cent.
MnO	0.18	per cent.
MgO	0.30	per cent.
SiO	2.40	per cent.
P	0.011	per cent.
S	0.009	per cent.

From a hand examination of the sample, and as judged by its fracture, the iron as compared to Pittsburgh iron would be graded as No. 3 foundry iron, that is, a close-grained iron, less open-grained than a No. 2 iron, and still less than a No. 1 iron. A photograph of the fracture of this iron, actual size, is shown in Fig. 1. Fig. 2 shows the same fracture as slightly over two diameters.

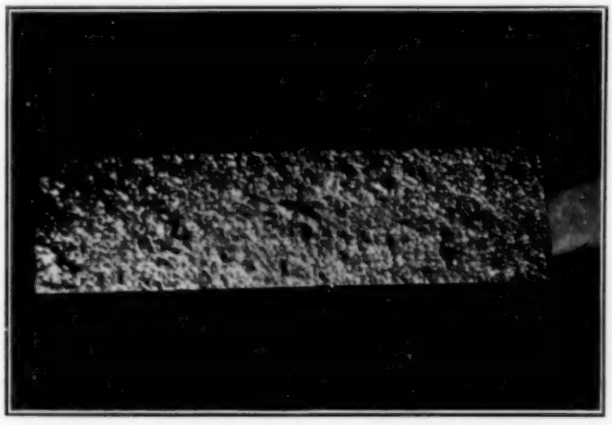


FIG. 1.—FRACTURE, ACTUAL SIZE.

A microscopical examination was made of a section of this sample, as shown in Figs. 3 and 4. In Fig. 3 is shown the sample unetched at 100 diameters. In the photo-micrograph the black areas represent the graphitic carbon and the white background the ferrite and combined carbon, if any of the latter be present. In Fig. 4 is shown the same area as is shown in Fig. 3, after etching with 10 per cent nitric acid and alcohol for about one and a half minutes. The junction lines of the ferrite grains are distinctly revealed, the small additional dark areas which appear representing pearlite areas. From an examination of these two specimens, etched and unetched, we are able to predict the constitution and physical properties that may be expected of such an iron. For example: In the first place we note that the carbon is practically all in the graphitic form, there being present not over 0.05 per cent of combined carbon. We also note the absence of the phosphide eutectic, which indicates that the iron is practically free from phosphorus. Also, by reason of the carbon being present almost entirely in the graphitic form, we are led to believe that the manganese and sulphur content of the iron are very low and the silicon content comparatively high. That this is true is shown by the following chemical analysis of the iron which was subsequently made:

Si	3.64	per cent.
S	0.00	per cent.
P	0.02	per cent.
Combined carbon	0.00	per cent.
Total carbon	3.58	per cent.
Mn	0.00	per cent.

The properties of such an iron, that is, one practically free from sulphur, phosphorus and manganese, have not, so far as we are aware, been studied due to the fact that pig iron of such purity is rarely produced. We are therefore only able to judge of the physical properties of such iron by noting the influence of the elements we find present upon iron *per se*,

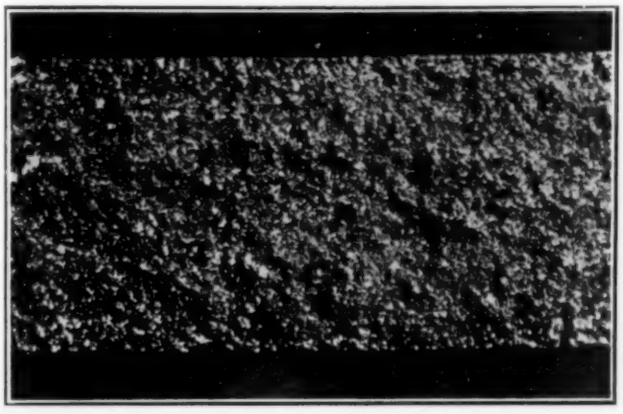


FIG. 2.—FRACTURE X2.

for, as has been shown by Howe, although one element may exert a very decided influence of itself upon the physical properties of iron, yet when in combination with others its effect is entirely different. In this case we have practically no other element present except iron, carbon, and silicon, and hence have only one element to consider, namely, silicon.

In order to study the influence of this element upon iron, Turner,* in 1885, prepared an iron by heating wrought iron and charcoal in closed crucibles. This was then remelted with a silicon pig, containing about 10 per cent silicon, in proportions necessary to yield any desired composition. However, the phosphorus content was in some cases as high as 0.32 per cent, and the manganese present in the most extreme cases was 1.95 per cent. Sulphur varied from 0.05 to 0.03 per cent. Therefore the irons compounded by Turner were in no instance as free from impurities as the iron under consideration, and hence we are unable to state by analogy what the principal properties of the latter should be.

The influence of silicon on the physical properties of cast iron has also been studied by W. J. Keep† and many others. Keep found that a small addition of silicon to white iron eliminates blowholes and produces soundness, while a further addition of that element gives the greatest crushing strength. He also observed that silicon in excess of that required to give the greatest crushing strength diminishes brittleness and yields the greatest transverse and tensile strength. This is, of course, due to the fact that silicon of itself has a tendency to cause whatever carbon may be present in the iron to assume the graphitic form. As might naturally be inferred from the manner in which the graphite is distributed throughout the iron, the strength of the matrix is necessarily less than would be the case if the graphite particles were not interspersed throughout the same.

Turner in his investigations of the influence of silicon upon the properties of cast iron arrived at conclusions similar to those of Keep. Turner obtained maximum hardness with an iron containing less than 0.8 per cent of silicon, maximum

*Journal Chem. Soc., 1885, p. 577.
†Am. Inst. Min'g Engs, 1888.

crushing strength with about 0.8 per cent, and greatest tensile strength with about 1.8 per cent. He also found that an iron possesses the softest and best working qualities when its silicon content was about 2.5 per cent. But it is to be remembered

FIG. 3.—MICROPHOTOGRAPH OF ELECTRIC PIG IRON.

that he performed his tests on iron containing appreciable amounts of manganese and phosphorus.

From the start it has been the aim of the Noble Electric Steel Company to produce an iron which would best meet the

FIG. 4.—MICROPHOTOGRAPH OF ELECTRIC PIG IRON.

requirements of the Pacific Coast foundries. In order to understand what these requirements are we may state that scrap cast iron is largely employed for making all grades of castings, and that the sulphur, phosphorus and combined carbon content of the same is very often so high as to impart very undesirable qualities to the finished product. For this reason the Pacific Coast foundrymen have been in need of an iron high in graphitic carbon and silicon and low in other impurities.

From the analysis given above it may be seen that the pig iron made in the electric furnace in California is of that nature. It is, therefore, apparent that if a foundryman using this iron understands the effects of phosphorus, manganese, sulphur, and especially silicon, upon the physical properties of iron he may by an intelligent mixing of this electric furnace pig iron with scrap cast iron, based upon actual analyses of the latter, obtain an iron in which the predominating physical property shall be the one most desired or a combination of physical properties best suited to meet the requirements.

Due to the fact that this electric furnace pig iron is practically free from phosphorus, sulphur, and manganese, while the irons used by Turner and Keep in their investigations contained appreciable quantities of other substances, the writers are of the opinion that a study of the former along lines similar to those pursued by Turner and Keep and Paglianti* will result in furnishing additional data on this subject. They are planning to conduct such an investigation in the very near future and hope to be able to supplement the data which has already been obtained on this subject by Turner, Keep and others.

*Metallurgie, 1911, 9, 217-230.

Harvard Metallurgical Laboratory.

On the Simultaneous Electrolytic Deposition of Copper and Zinc from Various Solutions Not Containing Cyanide

By M. de Kay Thompson.

The use of potassium baths for electroplating with gold, silver, zinc and other metals was discovered simultaneously and independently by Elkington and Ruolz in 1841.[1] Brass plating was first carried out by means of a cyanide bath by Ruolz in 1842,[2] though cyanide baths were not introduced into practice at first on account of their high cost.[3] At present brass plating is done only from a bath of potassium zinc and potassium copper cyanide, with a small excess of potassium cyanide. The cyanide brass bath is not entirely satisfactory, as it is slow and discoloration in spots is likely to appear after the surface has been polished and lacquered.[4]

Of the numerous different brass plating baths or brass plating methods that have been published or patented since the discovery by Ruolz of the use of cyanides for this purpose, there seem to have been only two in which cyanides were absent. The first was invented by Brunel, Bisson and Gaugain[5] in 1848 and has the following composition: 500 parts of potassium carbonate, 20 parts of copper chloride, 250 parts of ammonium nitrate and 40 parts of zinc sulphate. The volume in which these salts are dissolved is not given. In order to test this bath the above figures were divided by ten and these numbers of grams were dissolved (except for a small residue) in 450 cubic centimeters of water and electrolyzed with brass anodes and a platinum cathode. The temperature was 25° C. and the current density in the different experiments varied from 0.55 to 2 amperes per square decimeter. This bath proved to be of no value, for the deposits were in all cases copper, as was evident from their appearance.

The second method of brass plating without cyanides, referred to above, consists in using a sulphate solution of copper

[1] *Comptes Rendus* 13, 998-1021 (1841). The dates of the patents are Elkington, 8 Dec. 1840; Ruolz, 19 Dec. 1840.
[2] *Comptes Rendus* 15, 466 (1842).
[3] *Chem. Centralbl.* 1848, 506-508.
[4] Hogaboom, Trans. Am. Electroch. Soc. 19, 53-56 (1911).

and zinc with rotating cathode and anodes of copper and zinc on opposite sides of the cathode.[5] A high electromotive force from one dynamo was applied between the zinc and cathode and a low electromotive force from another dynamo between the copper and the cathode. The return wire from the cathode was common to both dynamos. By this arrangement it is said brass could be deposited on the cathode. It is rather difficult to see how brass could be obtained in this way, however, for in sulphate solutions of equal ionic concentration the potentials of zinc and copper are over a volt apart, and this cannot be very much reduced by making the zinc concentrated and the copper dilute. If the current density is increased greatly with the idea of reaching the potential difference between solution and cathode at which zinc would be deposited copper is certain to come down in a spongy form. Whether the current starts from a copper or a zinc anode only affects the composition of the bath and has nothing to do with determining which metal is deposited on the cathode at any given instant. A similar method to that of Jacobs was patented in 1858 by S. J. Goode for controlling the composition of cyanide baths,[6] though separate dynamos were not used. The current through the anodes was regulated by changing the depth to which they were immersed or their distance from the cathode.

It was thought that it would be an interesting subject for investigation to see whether some other than the usual cyanide bath could not be found from which brass could be plated and which would not have the disadvantages of the cyanide bath. This is the subject of the following paper.

In order to get a good electrolytic deposit of a metal the bath must not be too dilute with respect to the metal, but in order to have two metals whose electrolytic solution pressures are widely different deposit simultaneously, it is necessary that the ionic concentration of the less electropositive metal should be reduced to such a degree that the two potentials of the two metals in the solution should be nearly equal, that is:

$$\frac{RT}{n_1F}\log\frac{P_1}{p_1}=\frac{RT}{n_2F}\log\frac{P_2}{p_2}$$

where the subscripts refer to the two metals and the letters have the usual significance. It is not necessary that exact equality should exist, for the potentials at which the two metals deposit are reduced if they form an alloy at the instant of deposition, and also the concentration change taking place near the anode increase the potential difference between the cathode and the solution when the current is passing.[7] The only way in which low ionic concentration can be obtained with a comparatively large amount of the metal in solution is to have the metal in the form of a complex salt. If the ionic concentrations of both metals were equally reduced in any given complex not much relative change would be produced in the potentials of the two metals. Copper, however, has a much greater tendency to form complexes than zinc, and its potential is changed much more than that of zinc in the case of the cyanide solution[7] and probably in other cases also.

In attempting to find a bath from which copper and zinc can be deposited simultaneously the obvious thing to do was to try all the possible complex salts of copper. In order to measure out easily any desired amount of copper or zinc sulphate for making the complex salt solutions, a solution of copper sulphate containing one gram atom of copper per liter and another containing two gram atoms of zinc per liter were made up. The cathode used in the following experiments was polished platinum. The following is an account of the experiments with various complex salts.

1. The additions of glycerine, of sugar and of tartaric acid to the solution of a copper salt prevent some of the ionic reactions of copper, which indicates the formation of complex salts.[8] A bath was made up containing 0.25 gram atom of copper, 0.25 gram atom of zinc and 250 cubic centimeters of glycerine per liter. This was electrolyzed at room temperature with 2.6 and 1.6 ampere per square decimeter. The appearance of all of the deposits obtained showed them to be copper.

2. Sugar.—The effect of cane sugar was tested by electrolyzing a solution containing 0.5 gram atom of copper, 0.5 gram atom of zinc and 50 grams of sugar per liter. The current density was 3.3 amperes per square decimeter. This being very high, precipitated copper in a spongy state, but there was no evidence of zinc, as shown by the color.

3. Tartaric Acid.—A solution containing 0.5 gram atom of copper, 0.5 gram atom of zinc and 0.5 formula weight of tartaric acid was electrolyzed with 0.33 ampere per square decimeter. Only copper was deposited.

4. Glycocoll.—The addition of glycocoll to the solution of a copper salt produces a deep blue color and a very complex salt of the formula $Cu(NH_2CH_2COO)H_2O$, which is only partially precipitated by hydrogen sulphide. This seemed therefore to be promising, but on electrolyzing a solution containing 0.5 gram atom of copper, 0.5 gram atom of zinc and 100 grams of glycocoll per liter only copper was obtained. The current density was 3.85 amperes per square decimeter. At 60° C. the deposit was also copper, but of a lighter color. In cyanide baths the more free potassium cyanide is present the more the ionic concentrations of copper and zinc are reduced. It was not found, however, that 50 per cent more glycocoll added to the above bath had the effect of causing zinc to be deposited with the copper. This result is probably due to the formation of a very complex zinc salt, as explained above.

5. Sodium Potassium Tartarate.—A solution of copper was made according to Brown and Mathers[9] containing in 1000 grams of water 60 grams of copper sulphate (0.24 gram atom of copper), 50 grams of sodium hydrate and 159 grams of sodium potassium tartrate. On dipping metallic zinc into this solution it slowly becomes colored with a dark iridescent layer. A zinc solution was made from zinc sulphate and sodium hydrate containing 0.24 gram atom of zinc and 140 grams of sodium hydrate in 1000 grams of water. Equal volumes of these solutions were mixed and the resulting bath electrolyzed with a current density of 0.3 ampere per square decimeter. The deposit was smooth, adhered well and had a rich bronze color. The deposit was analyzed qualitatively and was found to contain some zinc. The same solution was electrolyzed with 0.6 ampere per square decimeter. The deposit was of a paler color and also contained zinc.

A bath was made from 150 cubic centimeters of the zincate solution and 50 cubic centimeters of the copper tartrate solution. On electrolyzing this with a current density of 0.3 ampere per square decimeter the deposit was similar to that obtained with the same current density in the solution made from equal volumes of the zinc and copper solutions, except that it was streaked in places with a granular deposit. This deposit also contained zinc. A great reduction in the total amount of copper in solution does not improve the deposit in this case nor in any other studied in this connection. Very little hydrogen was liberated on electrolyzing these solutions.

6. Thiosulphate.—On adding a solution of sodium thiosulphate to a copper salt the color changes to bright green and then to yellow, finally becoming colorless, with a large excess of the thiosulphate.[10] On electrolyzing a solution containing 0.4 gram atom of copper, 0.4 gram atom of zinc and 280 grams of sodium thiosulphate per liter, with a current density on the platinum cathode of 1.94 amperes per square decimeter, with brass anodes, a flaky bronze colored deposit was formed which did not adhere well to the cathode. This deposit was washed well with water and dissolved in nitric acid, except a small residue, which seemed to be sulphur. The amount of copper contained was determined electrolytically. The zinc was de-

[5] Charles B. Jacobs, *Jour. Am. Chem. Soc.* 27, 972-76 (1905).
[6] *Chem. Centralbl.* 1858, p. 350.
[7] Spitzer, *Zeit. f. Elektroch.* 11, 345-68 (1905).
[8] Abegg, Handbuch der anorg. Chem. 2, 547 (1908).
[9] *Jour. Phys. Chem.* 10, 39-51 (1906).
[10] Abegg, l. c., p. 557.

termined by precipitating with diammonium hydrogen phosphate as described in Talbot's "Quantitative Analysis." The results of a single analysis were the following:

Copper	50.5 per cent
Zinc	41.0 per cent
Deficit	8.5 per cent
	100.0

The deficit of 8.5 per cent is probably due to the residue which did not dissolve in nitric acid. The residue is probably also the cause of the bronze color. On allowing this solution to stand over night a muddy precipitate formed. It would therefore not be of any practical importance even though a good deposit could be obtained from it.

7. Phosphorous Acid.—The addition of phosphorous acid to a copper salt produces a complex copper phosphite anion.[11] A solution containing per liter 0.5 gram atom of copper, 0.5 gram atom of zinc and 50 cubic centimeters of phosphorous acid of 1.2 specific gravity was electrolyzed with a cathode current density of 1.33 amperes per square decimeter. A deposit of copper was obtained. On adding 10 cubic centimeters more of the acid to 100 cubic centimeters of this bath the deposit was the same.

8. Ammonia.—The deep blue color produced by adding ammonia to the solution of a copper salt is due to a complex copper ion, and it has been stated that the ammoniacal solution of copper and zinc is one of a number of others from which zinc and copper can be deposited simultaneously.[12] It was found that only copper is deposited from a solution containing 0.225 gram atom of copper, 0.225 gram atom of zinc and 660 cubic centimeters of strong ammonia per liter. This amount of ammonia precipitated and then dissolved the two metals.

9. Thioglycollate.—Salts of thioglycollic acid, $CH_2SHCOOH$, form complex salts with copper when added to a copper salt solution.[13] The potassium salt of thioglycollic acid was prepared by the method of Claesson,[14] which consists in adding one mol of monochloracetic acid to a solution of two mols of potassium sulfhydrate. The potassium chloride was filtered off and the filtrate treated with several times its volume of strong alcohol, which dissolves most of the potassium thioglycollate. After boiling off the alcohol the residue of white potassium thioglycollate was dissolved in water and added to 5 cubic centimeters of the copper solution, producing a bluish black precipitate which slowly turned white. This is the salt $(HOCOCH_2S)_2Cu_2$. This was filtered and dissolved in strong hydrochloric acid. The final solution contained 0.046 gram atom of copper, 0.36 gram atom of zinc, and was clear and colorless. The zinc sulfate was added to the copper glycollate without producing a precipitate. Electrolyzed with 0.07 ampere per square decimeter, only copper was deposited. On electrolyzing with a higher current density the solution became muddy.

10. Oxalate.—Copper oxalate forms a complex when dissolved in an excess of ammonium oxalate, in which the copper is in the anion.[15] A bath was made up containing in one liter 5 grams of oxalic acid, 55 grams of ammonium oxalate, and 15 grams of copper sulphate, as a similar solution, slightly more dilute, was found a good starting point for bronze plating.[16] When a zinc rod was dipped into this solution it took several minutes for it to become covered with a layer of copper, and when finally covered the copper was bright and adhered firmly. A solution of zinc was made from 12 cubic centimeters of the stock zinc sulfate solution and 40 grams of ammonium oxalate in one liter, corresponding to 0.024 gram atom of zinc per liter. A bath was made by mixing 150 cubic centimeters of the zinc solution with 50 cubic centimeters of the copper, corresponding to 0.015 gram atom of copper, 0.021 gram atom of zinc, 1.25 gram of oxalic acid, and 43.8 grams of ammonium oxalate per liter. With 0.17 ampere per square decimeter the deposit was spongy and apparently only copper. With 0.51 ampere per square decimeter the deposit was spongy and dark. It was washed and analyzed qualitatively for zinc; none was found present. With 1 ampere per square decimeter the deposit looked like brass in places and was found to contain zinc.

A solution was then made by mixing 25 cubic centimeters of the above copper oxalate solution with 175 cubic centimeters of the zinc oxalate solution and was electrolyzed with a cathode current density of 0.33 ampere per square decimeter. This gave a dark deposit. With 0.15 ampere per square decimeter the deposit was dark and easily wiped off.

Another solution made by mixing equal portions of the copper oxalate and the zinc oxalate solutions, thus containing in one liter 0.03 gram atom of copper, 0.024 gram atom of zinc, 2.5 grams of oxalic acid and 47 grams of ammonium oxalate. On electrolyzing with 0.28 ampere per square decimeter a deposit was obtained which first looked like smooth copper, but which later became granular. On increasing the current to 0.56 ampere per square decimeter, the deposit had a light brown color, and was found to contain zinc. The deposit had a similar appearance with a current density of 0.84. Hydrogen was liberated in all the electrolyses in oxalate solutions.

With this solution a rather curious observation was made. On repeating the electrolysis in the last mentioned bath with a current density of 0.56, the light brown deposit was not obtained, but the deposit seemed to be granular copper. On changing the platinum cathode for another, the light deposit was again produced. It is therefore evident that the cathode itself is of some consequence in determining the nature of the deposit. This experiment was repeated several times with the same result.

11. Pyrophosphate.—The nearest approach to success was with the solution made by dissolving zinc and copper pyrophosphate in an excess of sodium pyrophosphate.[17] Good deposits of zinc or of copper could be obtained from solutions containing only zinc or copper pyrophosphate. A solution was then made containing 0.054 gram atom of zinc, 0.054 gram atom of copper, and 108 grams of $Na_4P_2O_7 \cdot 10\,H_2O$ per liter. This sodium pyrophosphate includes that used to precipitate the metals and to redissolve them. On electrolyzing this solution with brass anodes and a platinum cathode with a cathode current density of 0.83 amp per square decimeter a brassy looking deposit was obtained. It was tested qualitatively for zinc, which was found present. There is a large polarization at the anode apparently caused by the resistance of a salt produced by the action of the current on the anodes, which does not dissolve readily at room temperature. If the solution is heated to 60 deg. C. this salt does not appear on the anodes. The deposit on the cathode is also good when obtained from a hot solution, but both in the hot and the cold solutions the deposit remains smooth and of a brassy appearance only for a short while. When the electrolysis is prolonged, the deposit becomes discolored, or granular. The solution around the cathode also becomes quite alkaline, as was noticed by the slippery feeling of the cathode on removal from the solution. If the cathode is rotated in order to stir up the solution in its neighborhood only copper is deposited by the current, which shows that the action of the current is partly responsible for producing the proper concentrations of copper and zinc ions near the cathode for depositing both metals together. It seemed that the alkalinity of the solution around the cathode caused the copper to form an oxide, thus spoiling the deposit. Various proportions of copper and zinc were tried without observing any improvement. The deposit is very much changed by a small change in the current density. The cathode frequently

[11] Abegg, l. c., p. 546.
[12] J. L. R. Morgan, *Jour.* Am. Ch. Soc. 22, 93-99 (1900).
[13] Abegg, l. c., p. 559.
[14] *Liebig's Ann.* 187, 113-126 (1877).
[15] Abegg, l. c., p. 549.
[16] Curry, *Jour. Phys. Chem.* 10, 515-20 (1906).
[17] Abegg, l. c., p. 546.

had an entirely different appearance in the center and on the edges, doubtless due to a slightly different current density at these points. In no case did the deposit continue to have a smooth surface when the electrolysis was prolonged. If the current density was low, only copper came down.

If a zinc rod is dipped into a solution of copper pyrophosphate, copper is deposited only very slowly, adhering firmly.

A deposit was obtained with a rotating platinum cathode frequently of a very light color like zinc, but on touching with a drop of nitric acid it turned copper colored immediately. It would seem therefore that this color is due to some oxide or hydrate of copper.

As the alkalinity of the solution around the cathode seemed to be the cause of the change in the appearance of the deposit, an attempt was also made to prevent the solution becoming alkaline by holding the pyrophosphate of zinc and copper in solution with pyrophosphoric acid. Free pyrophosphoric acid hydrolyzes slowly, changing to phosphoric acid, but there was a possibility that it would not do so when combined with these salts.

Pure copper pyrophosphate was made by adding an excess of copper solution to a hot solution containing 11.2 grams of $Na_4P_2O_7$, 10 H_2O, which forms 0.025 of a formula weight of copper pyrophosphate. This was washed with decantation with hot water until the wash water gave no test for sulphates. It was then filtered and transferred to an Erlenmeyer flask while moist. Then a fresh standardized solution of pyrophosphoric acid was run in from a burette until the solution became clear. A solution of zinc pyrophosphate in pyrophosphoric acid was prepared in the same way. The commercial pyrophosphoric acid was standardized against a solution of sodium carbonate with methyl orange as an indicator, taking into account that when the end point is reached only two hydrogen ions have been neutralized.[18] It was found that one formula weight of pyrophosphoric acid would dissolve 0.5 of a formula weight of zinc pyrophosphate, the final volume being 3.7 liters for the above amounts of acid and salt. In the case of copper 1.35 formula weight of the acid was required for 0.5 formula weight of copper pyrophosphate. The final volume here was 2.96 liters. With a rather high current density zinc could be deposited from the zinc pyrophosphate solution with the simultaneous evolution of hydrogen. Copper was easily deposited from its solution without evolving hydrogen. A solution containing both copper and zinc pyrophosphates was made by mixing these solutions, so that there was 0.15 gram atom of zinc and 0.15 gram atom of copper per liter. On electrolyzing this with brass anodes and varying current densities, the highest being 1 amp per square decimeter, only copper and hydrogen were deposited. The acid solution is therefore not as complex as when the salts are held in solution by sodium pyrophosphate.

On standing over night some of the zinc and copper pyrophosphate precipitates, showing that the combination with these salts does not prevent the hydrolysis of pyrophosphoric acid.

Summary and Conclusions.

Eleven complex copper salts were made by the following agents and were electrolyzed after mixing with zinc sulphate solution: glycerine, cane sugar, tartaric acid, glycocoll, sodium potassium tartrate and sodium hydrate, thiosulphate, phosphorous acid, ammonia, thioglycollate, oxalate and sodium pyrophosphate. The only solution which gave a smooth, bright deposit when electrolyzed for an hour or more was the alkaline sodium potassium tartrate solution. This deposit, however, had a bronze color.

The solutions from which appreciable amounts of zinc were deposited with copper were the alkaline potassium sodium tartrate, the oxalate and the pyrophosphate.

It was not found in any case that diminishing the total amount of copper in the solution improved the deposits. It seems that the complexity of the copper salt must be relied on to reduce the concentration of the ions sufficiently, while there must be enough copper present in the unionized state to furnish ions as rapidly as they are used up.

Massachusetts Institute of Technology.

[18] *Jour. Am. Ch. Soc.* 30, 735 (1909).

An Efficient Electric Furnace for High Temperatures.

BY D. F. CALHANE AND E. E. BARD.

In October, 1910, a description of a small electric furnace was given in this journal. As much interest was shown in the furnace from various quarters, it was decided to make it convenient for general use. In the past year and a half or so a great deal of work and investigation has been expended in producing the furnace that will be described in this article.

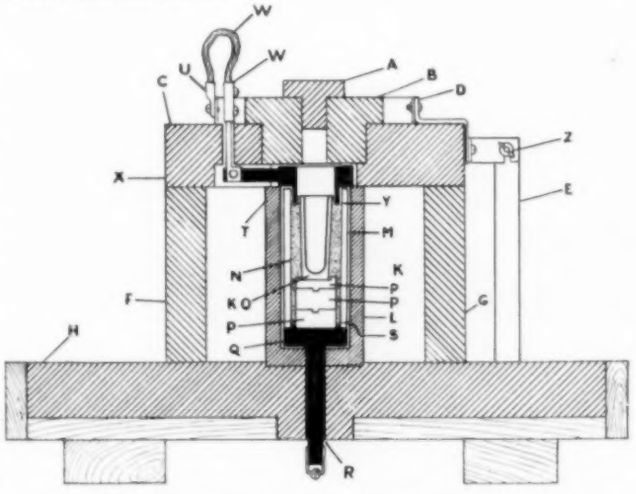
In the first place it was necessary to build the furnace in a way that would permit of easy assembling and dissembling.

Secondly, the use of refractories that would stand the exceedingly high temperature desired, in presence of incandescent carbon.

Thirdly, an electrical design that would give even distribution of current in the carbon core and sufficient resistance in the heating element to permit of using the furnace on the 110-volt circuit without transformers.

In what follows it will be shown how far these requirements have been met in the present form of the furnace.

The photograph gives a good idea of the general appearance of the furnace. It is 2 ft. long, 11½ in. wide and 33 inches extreme height.



SECTION OF FURNACE AS FINALLY COMPLETED SHOWING NEW TOP RING IN POSITION

In pursuance of the idea to make the furnace in such a form as to allow of quick and easy assembling and dissembling of the parts, it has been built upon a base *H* of cement composition, contained in a wooden frame *I* that rests upon wooden blocks *J*, allowing ready access to the electrode stem *R*. After considerable experimentation to find a composition that would have both good heat insulating qualities and a certain degree of strength necessary, a mixture of sand, Portland cement and magnesia-asbestos was decided upon. *E G* to fill the base mold *I*—7.45 kg. sand, 3.75 kg. cement and 1.25 kg. magnesia-asbestos were used These were mixed dry and then 6.0 kg. of water were used to convert the mass to a thick paste. This was placed in the mold *I* and allowed to set. Provision was made for a hole through the mass to permit of the insertion of the electrode stem *R*.

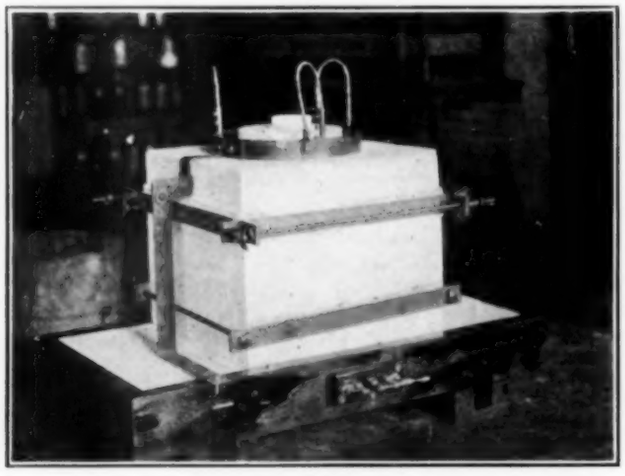
Upon this base stand four cement sides; the two longer ones shown in vertical section at *F* and *G*. Two end pieces not there shown. These sides are cast in simple wooden boxes, painted in order to prevent warping. The inner dimensions of side boxes are 12 in. x 6⅞ in. x 1⅝ in.—giving a volume of

134 cu. in. each, or 268 cu. in. for the two long sides. The inside of the moulds is greased with vaseline to permit of the removal of the casts. The mixture for these two sides is sand 4 kg., cement 1.6 kg., magnesia-asbestos .6 kg., water 3 kg. The moulds are put together with screws to allow of easy dissembling of the sides and removal of the casts.

The end pieces are cast in wooden moulds whose inside measurements are 7½ in. x 6⅞ in. x 1⅝ in., giving for the two a volume of 168 cu. in., requiring sand 2½ kg., cement .8 kg., asbestos-magnesia 35 kg. and water 2 kg.

The four sides stand upon the base and are held firmly together by two iron straps and binding screws, forming a rectangular space, the center of which is occupied by the fire-clay muffle *L*. This is made from two No. 40 furnace muffles manufactured by the Buffalo Dental Co., of Buffalo. The space *K*— *K* outside the muffle and within the walls is filled with shredded asbestos-magnesia for heat insulation.

The top *C* is also of cement mixture with three holes for the top electrode connections, and a large central hole for the cover *b*. This is cast in a wooden mold designed to give the proper contour to the cast. To fill the volume of 290 cu. in. a mixture of 4.5 kg. sand, 2 kg. cement, .75 kg. asbestos-magnesia and 3.3 kg. water are used.



ELECTRIC FURNACE.

In the central hole of the top, fits the cover *b*, that has a different composition owing to the higher heat it is to stand. The mixture used is sand 200 gr., cement 300 gr., asbestos-magnesia 300 gr., water 850 gr. and fire-clay 200 gr.

In the central hole of the cover is a cement plug *a* cast from a mixture of 40 gr. sand, 30 gr. cement, 20 gr. old fire-clay, 10 gr. new fire-clay, 40 gr. asbestos-magnesia and 100 gr. of water. All the sand used in these mixes passes 20 mesh screen. These castings as well as the top of the base, when completely dry, are given a heavy coat of water white for a finish.

Between the top *C* and the sides is placed an asbestos cloth gasket to help exclude air. Around the outside at the juncture of top with sides at *x* is fitted an asbestos cloth gasket sewed into one piece. This is bound air tight by four iron straps with tightening screws and rods, provided with thumb nuts. The photograph shows this arrangement. It is necessary to exclude air that will burn the carbon if allowed to enter.

By this arrangement it may be seen that the first requirement to make the furnace in a shape readily assembled and dissembled, has been met. Only seven cement parts have to be handled and these are put together and taken down in a very few minutes. Any one can make the moulds and cast these pieces with very little trouble.

Coming to the second requirement, the use of refractories that would stand the excessively high temperatures desired in presence of glowing carbon, the most difficult part of the problem was met. A great deal of experimentation has been done in the past twenty months and finally with the help of the Norton Company a line of alundum refractories has been developed that gives good satisfaction. The conditions are severe and put the alundum to very extreme tests. The alundum question is the backbone of the whole thing, and its wonderful properties make the furnace possible.

By reference to the cross-section figure it may be seen that the refractory portions consist of the crucible *O* that sets on three blocks *P*, *P*, *P*. The crucible is of pressed alundum 3¼ in. long, 1⅜ in. wide at top and 1⅛ in. wide at the bottom. It is 2¾ in. deep and will hold about 100 grams of molten steel of specific gravity 7.9. The three blocks *P*, *P*, *P*, are of tamped alundum of a special composition, to resist the excessive temperature at this part of the furnace. They are 1½ in. wide and ½ in. and ¾ in. thick. They are made in three sections to prevent cracking from unequal expansion.

Between the walls of the muffle *L* and the carbon core *N* is a tube of alundum *M* in two pieces 2 in. and 3⅛ in. long, respectively. This arrangement, in two pieces, is to avoid cracking from uneven stresses. The walls are 5/16 in. thick and bore is 1⅞ in. Another crucible of alundum is also used for certain work that is 5¼ in. long and 1¼ in. internal diameter. This sets on a block ¾ in. high.

These alundum parts stand excessive heat in contact with glowing carbon. The alundum parts formed of material compacted under hydraulic pressure give the ideal substance that is practically unaffected by incandescent carbon at 1900° C. It has taken months of investigation to determine the best composition and size of the various alundum parts now used in the furnace.

The third proposition, the electrical design, has been a difficult one. This for two reasons: In the first place it has been exceedingly hard to obtain even current distribution in the carbon core. Any inequality in this respect results in uneven heating of the alundum that causes cracking and at the higher ranges of temperature in burning out of the alundum by carbide formation. Again, it has been difficult to design the parts for current introduction and distribution, so as to make them simple, rugged and effective in operation. It has also been necessary to design the electrical part so as to get voltage drop enough across the furnace to work it direct on the 110-volt lighting circuit without transformer and only a simple rheostat to regulate the current. This has been obtained by a long, thin core of carbon, whose wattage density has had to be nicely adjusted so as not to exceed the endurance of the alundum parts at any point. After a great deal of experimentation and many changes in the electrical design success in this direction has been attained.

Referring again to the figure, p. 461. The current enters by *R*, the lead screwed into the electrode *Q* that rests on the bottom of the muffle. On the upper face of this is cut a circular lip *S*. In the inner space fit the blocks *P*, *P*, *P*, and down over the outer edge of the lip sets the alundum tube *M*. By this arrangement the current is evenly distributed all around from the upper face of the lip and through the carbon core *N* to the under side of the lip of the upper electrode *Y*. The means of exit from the upper electrode to the circuit is readily seen by an inspection of the drawing and general view of the furnace.

The voltage drop across the furnace can be varied within a range of 20 per cent by method of compacting carbon core *N* and the pressure exerted by the lip of the upper electrode on the carbon. This should be enough to give good contact, but undue pressure avoided. As an average of 10 heats, 80 volts across the furnace at 12½ amp. gave a kilowatt consumption of energy at full heat, melting point of platinum.

A block of fire-clay with a hole in the center is placed upon the crucible to conserve the heat. The furnace should not be

held at platinum temperature more than twenty minutes or half an hour, as the alundum does not endure so well. Alundum softens between 2000° C. and 2100° C. It is best to run the heat up to the m. p. of platinum as quickly as advisable and then pour the melt at once. The temperature of molten platinum is close to 1775° C. and the hottest parts of the furnace, especially the two upper alundum blocks are very near 2000° C. when the crucible holds melted platinum.

As a typical instance of the working of the furnace the data from a platinum heat are appended:

Time.	Amperes.	Volts.	Watts.	Temperature, Room Temp. at start.
3.21	8.5	60	510	
3.26	8.5	45	382	
3.38	10	45	450	dull red.
3.52	12	47	564	bright red.
4.02	16.5	53	874	
4.11	19	58	1102	white hot.
4.35	15.5	66	1022	
4.38	15	70	1050	platinum melts.

Twenty grams of platinum were melted in the above heat in an hour and 17 minutes, starting with the furnace at room temperature. Starting with the furnace warm from a previous heat, the same result as above was secured in one hour. With successive heats to melting point of platinum the crucible of pressed alundum endures very well. As many as twelve successive heats to platinum m.p. have not caused any marked effect on the material of the crucible.

Alundum tube *M* sustains the extreme temperatures very well, and will endure a number of successive heats to m.p. platinum without any appreciable deterioration. Its composition is of tamped alundum with low binder.

The two top blocks, *P, P, P*, have to endure extreme temperatures and will be made in the future of hydraulically pressed alundum, done in a steel mould. This material, as mentioned in the case of the crucible, is the ideal refractory.

The design here presented is the form most convenient for melting small amounts of very high fusing metals that can be done without carbon contamination and with good electrical efficiency. Other uses require slight modifications to fit the exigencies of the case.

The adaptation of this furnace to various uses is being studied in this laboratory and a larger size along the same lines, of 6 to 7 K.W. capacity, is being developed to give continuous operation. Patents are pending on the design here presented.

In conclusion, I wish to express my obligations to the Norton Co. for their kindness in developing the alundum parts; Also to Mr. P. A. Boeck of the same company for valuable assistance with the alundum question; also to Mr. Bard for his valuable suggesions that have led to marked improvements in the furnace design.

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Progress in Fuel Briquetting.—Although the United States is still far behind European countries, particularly Germany, in fuel briquetting, considerable progress has been made in the two last years. In 1909 the production of fuel briquets in the United States was 139,661 tons; in 1911 the production amounted to 212,443 tons. The main factor operating against the more rapid development of this industry is the low cost of raw fuel in the Middle West. The average price for fuel in the northern Middle States in 1911 was between $1.10 and $1.75 per ton. The cost of manufacturing briquets, exclusive of raw material, is about $1 per ton. Culm banks contain briquettable coal, and by-products from coke ovens can be used as binders.

The Arizona Copper Co. produced 1575 tons of copper in March, 1912.

The Elmore plant at Sulitelma, Norway, produced 1225 tons of copper concentrate in March, 1912.

A Zinc Retort with a Copper Bottom

BY F. L. CLERC.

The proposal to reduce and volatilize zinc from the surface of molten copper is so radical a departure from the lines along which zinc metallurgy has been developed that it is likely to impress unfavorably those most skilled in present practice.

It is the settled conviction of most zinc smelters that the metal must be reduced and distilled in a closed vessel of fireclay; that this vessel must be of small dimensions, and that a very considerable amount of high-temperature heat must be transmitted through its walls. With our present knowledge I believe that this conviction is in the main based on a solid foundation of facts. I wish, therefore, to show how closely the conditions within a copper-bottomed retort can be made to agree with the conditions in a fire-clay retort heated from the outside. It is admitted that the retort must be small, in at least one of its dimensions, in order that the heat can reach the charge to be reduced. In present practice the actual distillation is an intermittent operation, which is interrupted to charge and empty the retort. The period of firing may be divided into two very distinct stages, according to the work accomplished in each of them. In the "heating-up" stage the charge is dried; this must be done thoroughly in order to free the charge from volatile substances which would oxidize the distilled zinc if both were given off at the same time. How then can these two distinct stages be effected over a copper bath? How can the products of each of them be kept separate? And how can a closed vessel, heated from the outside, be formed with molten copper?

The answer to these sample questions must be somewhat categorical. The drying of the charge and the expulsion of troublesome volatile matter is done outside of the retort. The charge is ready for reduction when it enters the retort. The closed vessel for reduction and distillation is, of course, not wholly of copper. The bottom only is copper; the top, the sides and the ends are built of firebrick. The closed chamber, or retort, so formed, is heated from outside, as are all retorts, and therefore some of its dimensions are limited to the distance external heat can be made to pass into it. This composite retort is heated only through the bottom. For this reason its height is strictly limited. Its other dimensions are controlled by other considerations. Compared with cylindrical retorts, supported only at the ends and surrounded by flames, this may be thought an uneconomical manner of heating, but it has certain advantages. The length of cylindrical retorts is limited by the strength of the material of which they are composed.

In Silesian practice the retorts are made higher and longer, but at the expense of some heating surface, for they are supported along the bottom. There are other objections to these larger retorts, which are generally called muffles, and their use is restricted to localities which afford rather unusual conditions.

A copper-bottomed retort, with mechanical feed of a prepared charge, can be extended in length indefinitely.

As has been said, the heat comes through the bottom of the retort, but the source of heat is not below it. The copper extends beyond the retort, and this exterior projection receives the heat.

This is a familiar device to be seen in most chemical laboratories where a metal plate is used for heating. The desired temperature is obtained by moving toward or away from the hotter end of the plate. The plate is generally short, since the range of desired temperature is not great. The range of temperature desired for zinc reduction is even more restricted; consequently the limit of width of such a retort is soon reached. We now face the question, Can sufficient heat be passed through the bottom to affect reduction and volatilization of the zinc? In considering this question attention must

be given to both the temperature and the amount of heat required.

Heat Conductivity of Molten Copper.

The temperature of melted copper is well above the reduction point of zinc oxide by carbon. If, therefore, we keep the bottom molten the temperature will be sufficiently high. To get at the amount of heat which can be sent through the bottom we must know something about the conductivity of copper. It is generally known that this conductivity is considerably greater than that of any other of the common metals, and that all of the metals conduct heat much better than any earthy material. Measurements of the conductivity of solid copper, up to about 1000 deg. C., indicate that it diminishes rather slowly with rising temperature. This is what should be expected from what we know of the structure of commercial metals, all of which consist of an indefinite mixture of certain definite chemical compounds, with a ground-mass of so-called solid solutions. The solid solutions are unsegregated mixtures of two or more of the chemical compounds, which are more completely miscible at higher temperatures when all of them are fluid.

The flow of heat through solids is influenced by unhomogeneous structure, and its rate is likely to change at each separating surface or juncture. As a commercial metal approaches its temperature of fusion unknown changes occur in the arrangement of its constituents, and these changes must affect the heat conductivity. It is not safe to figure the conductivity of a molten metal by extrapolation from the formula of its conductivity as it approaches its fusion point. From heat phenomena of cooling metals certain temperatures can be noted at which definite changes in structure are known to take place. Before this segregation occurs the molten metal is more homogeneous. It is generally admitted that heat conduction in a molten metal is by metallic contact, where no force exists to produce convection currents. There can be little doubt that molten copper maintains its superior rank as a heat conductor, compared with other common metals when they are in the fluid state.

The freezing of melted copper in the ingot mold illustrates this.

Tests of the heat conductivity of clay materials show great variations. In general, it is probable that this conductivity is considerably greater at higher temperatures. But retorts are composed of artificial mixtures, of different clays in varying proportions, and there is a great range in the intimacy of the admixtures. It is, therefore, not safe to generalize on tests of isolated samples. It is difficult to measure accurately either the actual temperature within a zinc retort or the average temperature throughout a large heating chamber which contains several hundred retorts, and deductions made from such measurements are likely to lead to very erroneous conclusions. We can get a fairly adequate idea of the relative conductivity of copper and fire-clay by comparing the conductivity of copper at ordinary temperature with the conductivity of fire-clay corrected for high temperatures if we keep in mind that this ratio is subject to correction when the conductivity of molten copper is better known.

The conductivity of copper at ordinary temperatures is given as 0.92 cal., which means that this amount of heat will pass through a section 1 sq. cm. a distance of 1 cm. in one second for each degree of drop in temperature. The average conductivity of a specimen of burnt fire-clay, measured in the same way, between 0 deg. and 1300 deg., was found to be 0.0031 cal. Copper has, therefore, 300 times the conductivity of hot fire-clay under the above conditions.

Preheating the Charge in a Feeding Device.

Reference has been made to a preparatory treatment of the charge which should replace the drying and "heating-up" stage after it is in the retort.

It is not necessary to say much about the cost or possibility of briquetting the charge. When briquetting is successful in the treatment of much less valuable material, it offers no serious obstacle if anything is to be gained by its application.

A device is here shown (Figs. 1, 2 and 3) for heating the briquets by burning around them some of the carbon monoxide which passes out of the condensers. This heating is done in the mechanical feeder. No special merit is claimed for the arrangement shown, which can be varied indefinitely. It was designed to feed rather friable briquets and heat them to about 500 deg. C. It provides for a simultaneous feed through five hoppers, and has at least the advantage that it can be swung in its own plane and be raised out of the way of any work which must be done in repairs to the top of the furnace.

The gas, of high calorific power, is admitted into four sheet-iron compartments between the five spouts. Entrance of air and exit of products for each compartment is through holes in the sheathing. A common hood and a pipe through the roof of the furnace building carry off the burnt gas. By this preparatory treatment of the charge it is probable that ten hours of the work of an ordinary furnace can be dispensed with, and its capacity increased 12/7 times.

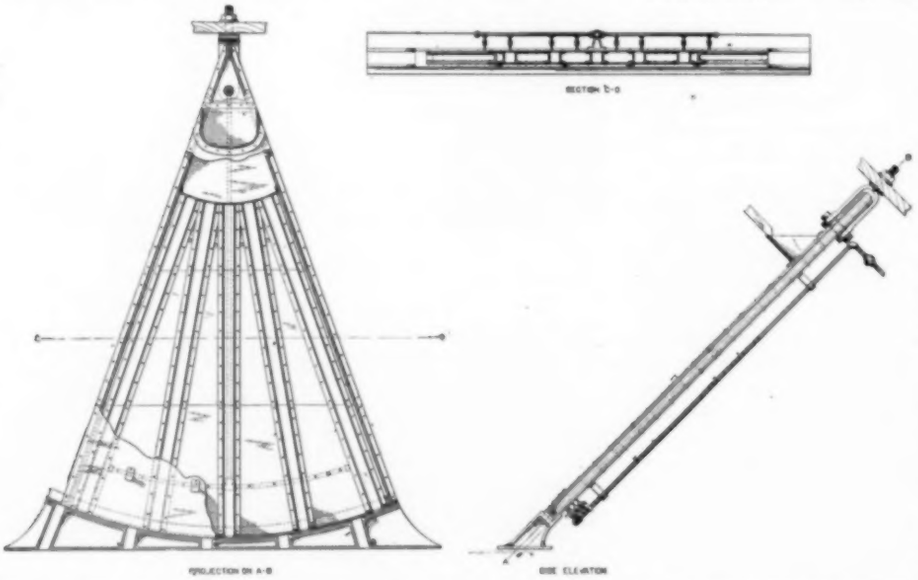


FIG. 1.—FEEDING DEVICE FOR CHARGING BRIQUETS OF ZINC ORE INTO COPPER-BOTTOMED RETORT. PROJECTION ON LINE A B OF FIG. 2.

FIG. 2.—SIDE ELEVATION OF FEEDING DEVICE.

FIG. 3.—SECTION OF FEEDING DEVICE, TAKEN ON LINE C D OF FIG. 1.

A compressed briquet of sublimed zinc oxide and carbon for reduction, measuring 4½ by 2 in., will weigh about 4 lb., and five will weigh 20 lb. Consequently the feeder will oscillate 100 times to deliver a ton of charge into the retort.

Construction of Furnace.

A general idea of the arrangement of the copper-bottomed retorts, and of how they are heated, can be gained from the

accompanying axiometric elevation shown in Fig. 4, and cross-sectional view, Fig. 5. I will here consider only two pairs of connected retorts having a heating chamber in common. The cut shows four pairs so connected. The heating chamber is centrally located between the retorts of each pair. Four such retorts, each with a bottom area of 15 by 74 in., would present a heating surface equal to four large cylindrical retorts, and would be contained in a block 15 by 8 ft. The bottom section of the furnace, which holds the pool of molten copper forming the bottom of the isolated retorts and the center well common to all of them, should be carefully built. It should have a casing of steel plates, should be strongly braced, and should have the best available heat insulation. This part of the furnace structure is almost indestructible. Above this level the superstructure of four retorts is raised to the height of about 7 in.

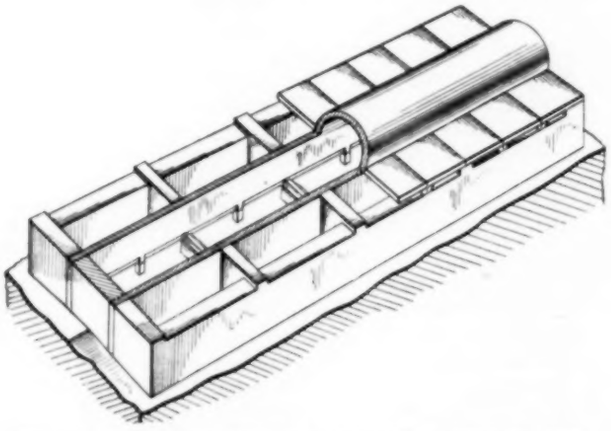


FIG. 4.—AXIOMETRIC ELEVATION OF COPPER-BOTTOMED ZINC FURNACE.

The outside walls, 15 ft. in length, are met at their middle points by a cross wall, which separates the first pair of retorts from the other pair. This transverse division wall extends entirely across the furnace block; but within the combustion chamber it is built up only to the top of the molten copper. The longitudinal division walls, which constitute partitions isolating each retort and separating it from the adjacent heating chamber, have the same height as the outside walls. They terminate below, just under the surface of the copper. They are supported midway in their length, at intervals of about 3 ft., by slender pillars of fire-brick resting on the bottom of the furnace. If the lower edge of these walls touches the copper when it is just melted, the expansion of the metal by heat will insure an effective seal. The buoyancy of fire-brick immersed in copper is considerable, and it will help support these flat lintels.

The outside walls will be cut down about 2 in. to form the slot through which the products enter the condenser. The top of the retorts will be formed of suitable plates spanning the 15-in. width of the retorts. These plates are tightly luted and are easily replaced. Through openings in these plates, at proper intervals, the charge is fed. The function of the full center-arch over the combustion chamber is to confine the heat and reflect it onto the copper to be heated. It has a span of 30 in. and supports little more than its own weight, which requires only a light section, but it should be of very refractory material. It could be moulded in two or three solid lengths to cover the full length of the furnace. This arch is covered with light insulating material, and should be encased in a jacket of steel plates. The slots through which the briquets are dropped can be built into this casing. With regenerative firing the hot gases should be brought from detached stoves, through moulded clay pipes jacketed with insulated coverings, and should pass out of the combustion chamber through similar pipes to the recuperators.

The flameless combustion of gas,* introduced by Prof. W. A. Bone, which consists in burning it within the pores of a very refractory diaphragm, and as it issues from them, would be a most effective method of heating the copper in the external well by direct radiation. In that case the arch with its insulating cover could be dispensed with, for the gas, under light pressure, keeps the top of the diaphragm cool and carries the heat downward to its lower face. The radiating surface can be brought close to the surface to be heated, and the top of the furnace will be left clear except for the feeding device.

Condensers.

The condensers are constructed separate from the furnace and are easily detached from it. They are supported and moved on wheels or rollers. Tracks should be laid parallel to the axis of the furnace and at right angles to it, so that the condensers can be moved expeditiously in either direction when it is necessary to replace one of them. When in position, they are secured in place and a good connection with the retort is effected by a sand joint.

The box of the condenser should consist of a sectional jacket of cast or rolled plates, some of them easily removable, so that access can be had to the interior. The lining should be of refractory insulating material, moulded, rammed and burned in place, as is usual with the lining of converters and ladles for hot metal. The gas escaping from the condensers can be utilized to some extent to effect this burning with gas-burning jets.

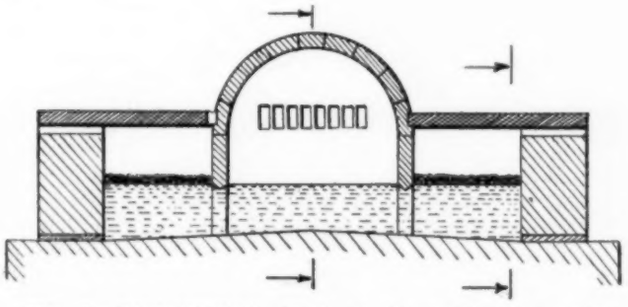


FIG. 5.—CROSS-SECTIONAL VIEW OF COPPER-BOTTOMED ZINC FURNACE.

It would be easy to deduce empirical dimensions for the condensers based on condensers now in use. Considerable variation will be found in these dimensions. The aim should be to make the condensers as large as practicable; that is, as large as it is possible to keep them properly heated, without being exposed to the danger of air being drawn in by changes of temperature within them. It is well to maintain a pool of melted zinc in the bottom of each condenser to equalize its temperature. The metal may be drawn off whenever the condenser is likely to become overheated, instead of emptying the condenser several times a day, as is done in Belgian furnaces, or once a day, as is the practice in Silesian works.

From the top of the last compartment of the condenser an adjustable iron pipe, fitting loosely into a socket, should lead to pipes connected with all the retorts for the purpose of conducting the gases to a common washer, in which the solid fumes are removed. The issuing gas is nearly pure carbon monoxide, without admixture with nitrogen.

With the dimensions here employed it will require forty-five briquets in a single layer to cover the bottom of each retort, making a charge of 180 lb. A second layer on top would add an equal amount without danger of crowding the disintegrated briquets over into the condenser. It is desirable to keep the retort well filled in order to minimize the cooling effect of feeding. Assuming then that a charge of 300 lb. is maintained in the retort, we can calculate how much its temperature is lowered every time 20 lb. of charge heated to 500 deg. C. is

*This journal Dec., 1911, p. 628; July, 1912, p. 409.

added to it. This charge is fed 100 times for each ton of charge reduced by the retort in twenty-four hours.

Contact of Charge with Heating Surface.

In both the Belgian and Silesian retorts the several particles of the charge remain in almost the same relative positions with respect to each other into which they were thrown in charging. Although there is some contraction in the charge as the zinc is worked off, this contraction is found to be objectionable, and as there is no way to replenish the charge while distillation is going on, it is customary to obviate contraction by mixing with the charge a large excess of coke or other inert material. The result is that the interior particles of charge do not get close to the walls of the retort. It is not probable that an intermittent feed of charge at a temperature of 500 deg. C. will cause greater fluctuations in the temperature within the retort than is inherent in any method of heating them. The hottest region in a copper-bottomed retort is at the surface of the copper. Here the zinc will be reduced and distilled. The prepared zinc oxide yields no considerable residue, and the ash of the reducing coal can be brought down to a minimum.

The lower briquets, by their weight, press closely against the copper. As they are consumed their places are taken by briquets fed from above in a way that avoids increasing the distance through which heat must be conducted to reach the particles of zinc oxide which are reduced. This average distance is evidently the length of an arc of the circle through the center of the sections through which the heat is conducted. This distance, 47.12 in., is eight times the average distance heat must be conducted to reach all the particles in a cylindrical retort 8 in. in diameter.

Reducing Capacity of Copper-Bottomed Retort.

We can now arrive at some approximation to the reducing capacity of a copper-bottomed retort by comparison with the Belgian retort, treating an equally rich charge.

Four copper-bottomed retorts having the same heating surface, working $\frac{12}{7}$ times as long, and having 300 times the conductivity of a fire-clay retort, but obliged to conduct heat eight times as far through their bottoms, should be equal to $4 \times \frac{12}{7} \times \frac{300}{8} = 257$ ordinary retorts. The comparison would be even more favorable if we should consider the conditions under which the retorts are heated in the two systems. It is with great difficulty that a uniformly high temperature can be maintained throughout a large chamber containing several hundred retorts; it is less difficult to maintain an even temperature within a specially designed combustion chamber of moderate dimensions. The effect of a higher temperature outside of the retorts is very marked. The excess of exterior over interior temperature in the retort is the potential which causes the heat to pass through its walls. The amount of heat which passes through them is directly proportional to this difference.

In the case of a copper-bottomed retort, the two endothermic reactions which occur close to the surface of the copper, namely, the reduction of zinc oxide by carbon monoxide and the reduction of carbon dioxide by carbon, will prevent its interior temperature from rising much above the fusion point of copper. Taking this to be 1080 deg. C., it can readily be seen that a temperature of 1280 deg. in the combustion chamber will produce twice the heat flow which is possible with a temperature of 1180 deg. in the combustion chamber.

The temperature which can be obtained in the combustion chamber is limited only by the resistance of its walls to heat; for, as I shall show, the consumption of fuel in heating ordinary zinc retorts is so great that a saving of fuel can be effected, even though the method adopted to heat the smaller chamber is not the best which can be devised. For example, in a recent publication Prof. Eng. Prost,* of Liege, gives a table of the performance of twelve typical modern zinc furnaces, which he credits to W. Borchers (*Métallurgie*, 1908, 522). From this I take as follows: The average fuel consumption of five Belgian furnaces per ton of zinc produced was 3.55 tons; for three Silesian and Belgo-Silesian furnaces it was 6.31 tons; for three Rhenish furnaces, 2.568 tons, and for one furnace using natural gas it was 2770 M^3.

Any saving in these amounts, when the costs and quality of the fuel are duly considered, may be regarded as satisfactory. The first object to be attained is to heat the copper bath effectively; when this is done, a more economical method of heating should be sought for.

Absorption of Zinc and Contamination of the Copper Bath.

Two other matters should be considered briefly: the absorption of zinc by the copper bath and the deterioration of the copper by contamination. Something can be learned on the first subject from the experience of brass founders. They are careful to immerse solid zinc in molten copper. The resulting brass remains fluid at a lower temperature, and since its specific heat is less, it holds less heat at this temperature. The cold zinc and its fusion absorb heat from the copper. On the other hand, the formation of the alloy is an exothermic reaction, of which the heat liberated is approximately known. If the loss of heat by radiation during the operation of compounding brass exceeds the sum of the heat given out by this reaction, and that no longer held by the specific heat requirements of the brass, the temperature of the brass will be lower than that of the copper to start with; otherwise the brass will be hotter than the copper. What would a brass founder answer to the proposal to make brass by pouring melted zinc on top of a ladle of hot copper, or, by passing a stream of mixed zinc gas and carbon monoxide over its surface? What would any one think of the proposal to form an alloy at a temperature at which this alloy is decomposed by heat? But it may be admitted that no chemical reactions are complete; that is, in the complexity of intermolecular action opposing reactions may take place simultaneously.

If, then, it is admitted that some absorption of zinc can occur, the ratio of the weight of zinc absorbed to the weight of zinc which escapes absorption is directly proportional to the relative velocity with which it passes away from the surface of the copper in opposite directions, this being the only place at which zinc and copper are in contact. Above the copper the zinc has the velocity due to its own expansion in passing from a solid to a gaseous state and due to the expansion of an equal volume of carbon monoxide simultaneously produced. This velocity must amount to many meters per second, and is very great compared with any velocity of diffusion that can be assumed to be possible through the body of the copper.

The danger of contamination of the bath by impurities contained in the sublimed zinc oxide is slight, both because of the small amount of deleterious matter it contains and because the impurities which it does contain are highly comminuted. The separate particles of these impurities, scattered through very light flakes of oxide of zinc, have no chance to come together; they will be liberated by the volatilization of the zinc in isolated particles, which are readily carried away as fume. The reducing coal is the greatest source of contamination, and its quality is a matter of prime importance. Any impurity which can be volatilized or carried away by the gases, before the region of greatest heat is reached, will have little chance to come in contact with the copper.

Conclusion.

In conclusion, it is not out of place to call attention to the fact that recent industrial progress has been due largely to two

*Cours de Métallurgie des Metaux autres que le Fer, Paris and Liege, Ch. Beranger, 1912.

underlying influences, the strength acquired by co-operation, and the efficiency attained by specialization. In the domain of hand labor, co-operation involves the concurrence of numerous individual wills acting for a common object. Under these circumstances specialization of function stands for individual skill. When machinery supplements labor, having no will of its own, it must be directed by intelligence. Corresponding to a concurrence of wills there must be a consistent system or policy evolved in order to reach the desired end. Special tools are elaborated which are highly efficient in special work, and intelligent direction of them is to be found in the mechanical development of metallurgy, which concerns the shaping and use of metals. Modern progress in these directions is due largely to the extent to which co-operation and specialization have become effective. Analytical methods of chemical laboratories have been successfully introduced in the development of metallurgy on its chemical side, which may be distinguished as manufacturing chemistry. Between mechanical and more strictly chemical metallurgy there remains a vast field of metallurgical industries which have felt these modern influences and adopted these modern methods to a greater or less extent. I believe it will be found that those special forms of metallurgy which have shown the greatest progress are those in which co-operation and specialization have been carried furthest. A special tool does a special work well; a special process should do one thing well. Metallurgy begins in separating a metal from its associates in nature. In the laboratory the elements and compounds are separated into groups. Isolation is reached by gradual stages. Desired compounds can then be formed. In the metallurgy of iron, copper, gold, silver and aluminium the distinct stages are clearly defined and they are perfected separately. I think a notable advance in zinc metallurgy will result from a proper recognition of the advantage of preparing its raw material before it is submitted to the complicated and expensive operation of reducing and distilling the metal it contains. Cost of a product does not depend on the number of operations required to produce it. It does depend on the simplicity of these operations and the ease with which they can be controlled and accomplished.

Co-operation when machinery is employed, and when numerous processes are required for the separation of metals associated together in nature, can only be secured under intelligent direction. From the fact that several metals are found together it results that a company organized to extract one metal must handle several others.

Zinc usually is found with lead or copper and with gold and silver. In present practice one of the metals in the ore is sacrificed more or less to obtain the others. Zinc is troublesome to work with other metals, and the immediate object is to get rid of it. When most of it is driven off, the residue from this treatment, whether it be a slag or a matte, can be worked for the extraction of the other metals which it contains. It is well known to metallurgists that the zinc driven off carries with it in the furnace gases notable amounts of more valuable metals.

I think it is safe to believe that practically all of the gold and silver contained in sublimed zinc oxide prepared for spelter production will be caught and held by the copper in a copper-bottomed retort or by the condensed zinc metal. This belief is based on the known affinities of these metals. The collection of an amount of the precious metals, too small to enter into the valuation formula, perhaps even too small to be detected by commercial analyses, may amount in the aggregate to enough to pay for the renewal of the copper bottom and the cost of refining the copper. It would appear that zinc smelting should be carried on as one department of a larger industry by which all of the valuable constituents of the ore treated would be utilized, so that by-products would be equivalent to material at intermediate stages of the system of production.

Estes Park, Col.

Metallography and Metallurgy at the Bureau of Standards.

By George K. Burgess, Sc. D.

Among the functions of the Bureau of Standards is the determination of physical constants and the properties of materials when such data are of great importance to scientific and manufacturing interests. Although there are innumerable physical constants and properties of materials that are not yet known with the accuracy oftentimes needed in the sciences and arts, a selection is most naturally made from among those problems for which on the one hand the need for their solution is presented in some concrete form, and on the other hand from among those problems which in themselves are evidently fundamental and form a basis for many applications.

A glance through a standard treatise such as Professor J. W. Richards' "Metallurgical Calculations" will convince anyone of our deplorable lack of sufficiently exact knowledge of many of the constants and fundamental properties of materials necessary to a proper carrying out of many of even the more usual operations in metals alone, to say nothing of the other metallurgical products. For example, at the temperatures at which they are usually worked, for most metals we do not know either their specific heat, heat of fusion, thermal or electrical conductivity, porosity to gases, thermal expansion, mechanical properties, of which there are so many, or their radiation properties, nor for any one metal do we know all of these well enough even for a rough calculation of the efficiency of a projected new operation.

The Bureau of Standards receives almost daily requests for more exact information in this field, but cannot pretend, with its present limited staff and equipment, to keep abreast even of the most urgent demands. As has been pointed out by others, the United States lags behind Europe in an aggressive attack from the scientific side, which forms the basis for all rational development, of the problems underlying the metal industries and on which their ultimate economy depends.

By studying just such fundamental problems of interest either directly or indirectly to the metallurgist, the Bureau of Standards hopes to be able in some measure to be as useful to the scientific and industrial world in general and to the metal industries of America in particular, as are the corresponding foreign national physical and testing laboratories such as those of England, France and Germany.

To cite some of the metallographic work of but one of these, the English, we may mention the comprehensive study of the iron-carbon system and of the heat treatment of high-speed steels, and more recently the mechanical properties of steel at high temperatures and a very elaborate study of aluminium alloys, some of which find application in aeronautical and naval construction.

In the field of metallography and physical metallurgy there are, of course, individuals in this and other countries who are working independently and producing admirable results; nevertheless many urgent problems require complicated and costly equipment, and others are so elaborate that the association of a group of persons becomes imperative for a complete solution within a reasonable time of all the phases involved; and such cases fall naturally to the lot of the laboratory which has facilities for attacking simultaneously the various aspects of the problem such as physical, chemical, metallographical, and mechanical, and on a scale, when necessary, of sufficient magnitude involving large units to work with, that an individual would in most cases not care to consider.

The testing and research laboratories of some of the larger manufacturers also produce, oftentimes, results of the greatest scientific interest and practical importance, but usually they are mainly interested in attaining some definite technical goal, so that in their general work, when available, requires supplementing and generalizing to make it most useful. This is particu-

larly true for investigations of the metals and their alloys; the temptation for a commercial laboratory to discontinue an investigation when a given material having the sought-for properties is attained is too great to be resisted. For example, an electrical firm seeking a high resistance, high melting-point alloy of small temperature coefficient, will probably drop the problem when a suitable alloy is found, regardless of other equally valuable (to someone else perhaps) properties that may have shown signs of developing during the research.

For all those reasons it is clear that a national physical and materials laboratory has a very definite rôle assigned to it in its relation to the progress of metallurgical science on the one hand and to the metallurgical industries on the other.

The question may be raised as to the respective provinces of the Bureau of Mines and the Bureau of Standards. The former occupies the domain between the Geological Survey and the Bureau of Standards, and in the words of the report of the House Committee on Mines and Mining:*

"Nothing in these provisions (i.e. the bill H.R.17260 amending the Act establishing the Bureau of Mines) would authorize or permit the Bureau of Mines to extend its investigations beyond 'mining' operations so as to duplicate the geologic and topographic work of the Geological Survey; nor will anything herein authorize the Bureau of Mines to extend its work to cover the investigation of finished manufactured products, which come properly within the scope of other branches of the Government service."

Briefly, then, the Geological Survey deals with materials in the ground, the Bureau of Mines with getting them out of the ground and refinement from the ores, and the Bureau of Standards with the properties of the manufactured material. Another well recognized phase of the work of the Bureau of Standards is, of course, the investigation of testing methods and the certification of measuring instruments of all sorts.

The Bureau has for some time been engaged on problems of interest to the metallurgist. One of the most important is the preparation and analysis of standard samples of irons, steels, alloys, and ores which are distributed broadcast to works chemists and others interested, for a small fee, and serve as a basis for obtaining uniform analytical results in all localities by checking both the metallurgical chemist and his methods. Incidentally new and better methods of analysis have been devised and described for some of these materials.

Samples of standard substances, such as naphthalene, sucrose, and benzoic acid are similarly distributed for the standardization of combustion or fuel calorimeters. The preliminary study and preparation of these calorimetric combustion standards have entailed very elaborate investigations which are still in progress and in the course of which new methods and apparatus have been devised. A study of the calorimetric and thermal properties of various gases is also under way.

A very troublesome problem of the greatest practical importance, which is being investigated with thoroughness and on a large scale, is the resistance of steels to electrolytic and corrosive action. Some extremely interesting data have been obtained which will be published shortly.

The mechanical properties of the metals, especially of steels suitable for structures, have received considerable attention, although this work has been but recently organized, and with the completion of the new machines now being installed at Washington and Pittsburgh, the Bureau will be equipped for testing of large members, as well as for the usual test sizes.

A comparative experimental study of the methods of determining hardness has been made, a modification of the Brinnell test suggested, and several investigations on the correlation of the physical and mechanical properties of steels are under way, having as a partial object the simplification of the test.

Some of the thermal properties of the metals have been studied with particular care; for example, a considerable number of melting points have been determined with exactness for use in defining the temperature scale to 3000° C. An investigation of the reproducibility of some of these standard temperatures by means of metals from various sources has also been made, and a new method devised for the determination of the melting points of microscopic samples. A study of the methods of obtaining cooling curves has been published.

Sir Robert Hadfield has placed at our disposal several hundred samples of steel for the determination of melting and transition points, hardness, etc. These steels, covering a very complete range of compositions, are of particular interest as many of their mechanical, magnetic, and electrical properties have already been studied elsewhere.

A thermal analysis has been made of a series of standard steels for Prof. H. M. Howe in conjunction with his thermo-microscopical study of grain size. The results of this investigation have confirmed our previous opinion of the desirability, in view of the existing confusion on many points, of attacking anew the iron-carbon series of alloys from the thermal standpoint, using only pure components, i.e. eliminating all impurities, and studying one by one all the factors that enter into the location of the transition points on heating as well as on cooling, such as methods of measuring the phenomena, rate of temperature change, atmosphere, impurities, size of sample, etc., etc. Starting with pure iron, some interesting and unexpected results have been obtained which will be published shortly.

It is planned to attack in a similar way other series of alloys of practical and scientific importance, and to determine for these alloys such of their mechanical and physical properties as may be of interest as well as their constitution, microstructure and heat treatment. For some time past the Bureau has been equipped for the examination and heat treatment of small samples including the taking of heating and cooling curves in vacuo, in air or other gas. The equipment is being extended to permit the handling of larger test pieces.

Of considerable practical importance in the working of hot metals, is the ability to determine their temperatures correctly with optical and radiation pyrometers. Such corrections have been published by the Bureau for pure copper and copper-oxide, the experiments for iron-oxide have just been made as well as some preliminary work done on liquid steel, aluminium and oxides. Nickel, special steels and other substances of practical importance are to be studied, the emissivity for each substance throughout the visible spectrum being also determined.

It is hoped shortly to be able to take up some of the high-temperature electro-metallurgical problems, particularly those of interest in furnace design and operation such as electrical and heat conductivity and temperature coefficients of metals, electrodes, furnace materials, and charges, carrying the measurements to the actual working temperatures. As to the non-metallic materials of interest to the metallurgist, a paper is in press on the melting points of fire-clays covering representative U. S. samples. A study has also been made of the load-carrying capacity of such bricks at high temperatures, and the determination of the refractory qualities of such material forms a part of the routine work of the Bureau.

Another field of usefulness of the Bureau is aid furnished in the drawing up of specifications in connection with the technical societies interested and after experimental study. In the metallurgical field there is room for a great deal of work here. Again, there is the question of nomenclature, for example, of bronzes and brasses, which has been raised before the Institute of Metals, and with new alloys appearing almost every day, some rational system of naming becomes imperative.

In conclusion, it may be said that it is the policy of the Bureau of Standards not to concern itself with those problems that are best treated in private laboratories nor is there any desire to encroach on the field of the consulting engineer or ordinary testing laboratory.

Washington, D. C.

*House of Representatives, 62d Congress, Report No. 243, 1912.

The Extraction of Vanadium from Carnotite and Vanadiferous Sandstone Concentrates

BY SIEGFRIED FISCHER.

Within the last decade numerous attempts have been made to extract vanadium from carnotite by a process that would be commercially feasible. These efforts have been based on two principles: first, dissolving all the metal constituents of the ore by means of acids and then separating uranium from vanadium by precipitating the latter; and, secondly, changing the insoluble vanadium into the soluble form by means of caustic alkali, salt and heat, and then leaching and precipitating the extracted metal.

I believe that, up to the present time, only the second of these two principles is of commercial importance. Patents have been granted on the acid treatment, but none of the processes has been successfully adopted in practice. While their theory is very plausible, they encounter various difficulties in practice, some of which will be discussed later.

Extraction by Acids.

The acid treatment is susceptible to several modifications. One method consists in making a direct extraction of uranium and vanadium from carnotite by means of a solution of sulphurous acid. Later the solution is boiled, whereupon the uranium is precipitated as basic sulphite, and subsequently calcined to uranium oxide. The vanadium remains in solution and is precipitated by means of caustic lime.

By another method the ore is ground and treated with dilute sulphuric acid, which will dissolve the uranium, vanadium, copper and iron contained in the ore. The solution containing these metals is reduced with sulphur dioxide, producing ferrous and vanadous compounds. A calculated quantity of pulverized limestone is then added until the metals would just begin to separate, calcium sulphate being formed. The solution is again filtered, and the precipitation of the metals completed by boiling with pulverized limestone. The precipitate is a mixture of basic sulphates and carbonates of uranium, vanadium, copper and iron. If it is desired further to refine this product it may then be treated with sulphurous acid solution as indicated in the first method mentioned above.

In practice, however, a process of this type will encounter many difficulties. When we consider that nearly all precipitated metal vanadates are gelatinous, and that they will hold varying amounts of other metals as impurities, the washing of such a precipitate becomes a problem in itself. In case the vanadium is precipitated as calcium vanadate, the amount of calcium sulphate formed will handicap the production of a clean precipitate of vanadate. This will necessitate further steps for the elimination of this substance, and more time will be required to produce a clean precipitate, all of which will increase the cost of production. Cost is an important item in these chemical processes, especially in view of the fact that the largest producer of vanadium encounters none of these difficulties in treating its ore.

Extraction by Alkalies.

The second class of processes for extracting vanadium from raw ore may be typified by the method patented by Mr. W. F. Bleecker.[1] In my opinion, Bleecker's process is, at present, the only one which promises commercial success. Briefly, it is as follows: The ore is mixed with caustic alkali and salt, and subjected to a roast at a temperature not above that of low red heat. If heated above this point the vanadium becomes insoluble again. The roasted product is then leached with water, the alkaline vanadate passing into the solution. After filtering, the vanadium is precipitated by the addition of a salt of copper, lead or iron, and the desired metal vanadate obtained. This product can be converted without difficulty into the corresponding vanadium alloy.

[1]Patent, No. 1,015,469, issued Jan. 23, 1912, to W. F. Bleecker, Canonsburg, Pa.

I have successfully applied this process to six different lots of vanadium ore. Different ores of carnotite, such as those contained in sandstone or clay, were mixed with equally good results. The extractions were high; and from two of the samples the residues showed only a trace of vanadium after leaching with hot water.

After considerable work for the last two years the author has come to the conclusion that the only successful means of extraction of vanadium is by means of the caustic alkalies, namely, sodium and potassium hydroxide. Many chemicals have been tried, both in the wet and dry way, but none of them gave satisfactory results on the *crude ore*. Tests were made with all of the mineral acids, as well as with some of the simpler organic acids, but without material success. The acids would attack the uranium as well as the vanadium and this means of extraction had to be abandoned. Potassium cyanide and cyanamide were tried, but without success. In the dry way, fusions were made with phosphates, fixed alkalies, and mixtures of these chemicals with carbon in an effort to render one element soluble; but while some experiments gave extractions as high as 65 per cent, none of the methods seemed wholly satisfactory.

It was never possible for me, with pure alkali alone, to obtain satisfactory results on the *crude ore*, using the wet method, even on boiling. On using commercially prohibitive quantities of alkali nearly complete extractions were obtained. The reason for the excessive consumption of caustic alkali probably was due to the reaction between that chemical and the silica in the ore. This led me to consider the concentration of carnotite and similar ores in the hope of eliminating the troublesome silica and securing a more suitable mineral product for treatment.[2]

Advantages in Treating Concentrates.

On applying the caustic treatment to concentrates improved results were immediately apparent. Much smaller amounts of the chemical were necessary in order to obtain good results, than had been the case in treating crude ore. The highest amount of caustic used was in the ratio of one ton of alkali to five tons of concentrate; and as the ratio of concentration was 1:3, the consumption of caustic alkali per ton of original ore amounted to only 135 lb. With this treatment I obtained an extraction of 93.44% vanadium.

Naturally the consumption of alkali must be referred to crude ore, and not to concentrate, in order to compare the process with those which work on crude ore. The following tests will show the importance of concentration. An ore carrying 3% vanadium was treated with caustic soda in the ratio of 100 parts crude ore to 20 parts alkali. After leaching, the residue still gave a strong reaction for vanadium. One hundred grams of concentrate from the same ore, carrying 8.97% vanadium, was treated in a similar manner with 20 grams of caustic soda. The residue after leaching gave only a slight reaction for vanadium. The concentration ratio was 3:1. In order to obtain 100 gm. concentrate, I was obliged to use 300 gm. crude ore. Basing the alkali consumption on crude ore treated, it is not excessive.

The foregoing tests showed, also, the important role which silica plays in the alkali treatment. If the liquor obtained from the crude ore extraction is acidified, a dense precipitate will be observed, which is readily identified as silica. A similar test on the extraction liquor from the treatment of concentrates will show scarcely any cloudiness. Even if a small amount of silica should be present, it would not be a menace to the further steps of the process. As it is apparent that this method of attack is not commercially applicable to crude ores, and is successful in treating concentrates from which the bulk of the silicious material has been removed, I have applied for a patent on the extraction of vanadium from the concen-

[2]This journal, June, 1912, page 356.

trates of carnotite and vanadiferous sandstone by means of caustic alkali.

Details of the Process.

The process comprises three distinct steps:

1. The conversion of insoluble vanadium into the soluble form.
2. Leaching and separating vanadium from uranium.
3. Precipitating and drying the vanadium compound.

The dried product is then ready for reduction to ferro-vanadium, and the uranium residue is in marketable shape.

The procedure is as follows, reference being made to the accompanying diagrammatic flow-sheet. The necessary

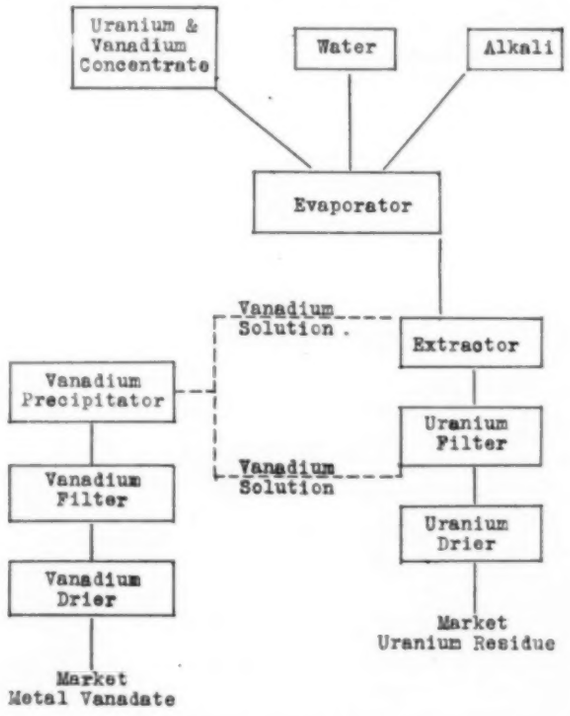


FIG. 1.—DIAGRAM SHOWING TREATMENT OF CARNOTITE CONCENTRATE.

amount of alkali is dissolved in water in the evaporator, and concentrate is then added in proper proportion. The whole is mixed to a pasty consistency in order to insure intimate contact of the alkali and mineral, and then evaporated to dryness, finishing at a temperature ranging between 200 and 300 deg. C. In some cases it may be necessary to repeat this step by the addition of water and subsequent evaporation. Tests will indicate whether the attack of the alkali has been complete. The dry mass is now ready for extraction with water, and is transferred to the leaching tank, or extractor.

The leaching tank is steam-jacketed, and is provided with a stirrer to thoroughly disintegrate the mass and produce a slight friction or abrasion of the solids. The vanadium passes into solution, and the latter is pumped to the vanadium precipitation tank. Here the solution is neutralized with sulphuric acid, after which iron vanadate is precipitated by the addition of ferrous sulphate. The mixture is filtered, and the precipitate dried and reduced to ferro-vanadium in the electric furnace or by the Goldschmidt process.

The residue from the original leaching step is filtered and washed, the liquor passing to the vanadium precipitator after it has reached a suitable degree of concentration by repeated use as wash liquor. The uranium residue is dried and marketed, being in excellent condition for further treatment on account of the previous removal of the silica by concentration.

The process permits the use of a minimum amount of chemicals, requires but little labor, is easily manipulated, and can handle large amounts of concentrates. It is practically continuous, that is, successive lots of ore can be kept continually moving through the various steps of the process.

The completion of the reactions can be tested by taking samples from various tanks, thereby controlling the process. Tests can be made to determine the efficiency of the alkali reaction, the precipitation of the vanadium and the washing of the various precipitates and residues. Ordinarily intelligent labor can be taught to make these tests, so that technically skilled labor will not be required. It would be a good scheme to have a testing equipment for the concentration and extraction of ores in 50 or 100 lb. lots. This would give the best preliminary information regarding the dies to be treated.

To obtain the highest efficiency, various conditions must be taken into consideration. The concentrates treated must be of uniform character. This can be accomplished by mixing various ores before concentration so that the proper ratio of concentration can be maintained. By careful analysis of the different ores, any difficulty in this respect may be avoided.

Another factor which must be considered is the loss of vanadium by washing. While the amounts which may be lost from time to time may seem slight, they will amount to considerable in the course of a year's operations. The author has given quite some time to the study of this point and may publish the results of his experiments in a later paper.

Comparison of Results.

Utah and Colorado carnotites have been treated by this process, both in the crude condition and as concentrates. The following figures will show the advantage to be gained by treating concentrates instead of crude ore.

Utah Crude Ore: Assay, .3% vanadium; treatment, 100 parts ore and 20 parts caustic alkali. Extraction liquor contained large amount of silica. *Extraction*, 65.3%.

Utah Concentrate: Assay, 8.97% vanadium; treatment, 100 parts concentrate and 20 parts alkali. Extraction liquor showed no turbidity on acidifying, and the precipitated iron vanadate contained no aluminum. *Extraction*, 94.66%.

Colorado Crude Ore: Assay, 5.59% vanadium; same treatment as above for Utah crude ore. Extraction liquor contained much silica. *Extraction*, 67.03%.

Colorado Concentrate: Assay, 16.34% vanadium; same treatment as for Utah concentrate. No turbidity in extraction liquor on addition of acid, and no aluminum in the iron vanadate. *Extraction*, 92.98%.

In all, about 20 different ores were tried, and none of them gave less than 87% extraction when treated as concentrates. Smaller amounts of alkali were tried in some instances, with satisfactory results. The process never required more than 1 part of alkali to 5 parts of concentrate. Many samples were tested for copper alone, and while some of them showed the presence of this element in the crude ore, it was scarcely noticeable in the concentrate, and the resulting iron vanadate showed no trace of it.

The important part of the process evidently is the first step —converting the insoluble vanadium into the soluble form. The water added at this point is not a leaching agent, but a means of carrying the alkali and bringing it in contact with every particle of carnotite. It is essential that the alkali should be uniformly distributed throughout the mass. As the solution becomes more concentrated, the action of the alkali is more intense, and I have found that in some cases it was unnecessary to perform the final drying operation at a high temperature. The alkali in this finely disseminated form evidently has a better opportunity to act than if it were mixed dry with the ore. The latter method would cause lumps to form, and give less efficient reaction between the ingredients. It was found by experiment that if the mass was kept moist and at a temperature of 100 deg. C., the change from insoluble to soluble form of vanadium was nearly complete. Whether this mode of treatment would be more economical than the

evaporation and calcining, remains to be seen. A factor which naturally comes into consideration is time.

We may now consider the merits of this process. It is apparent that by the original concentration of the ore a large quantity of undesirable material is removed, thus increasing the probability of getting a pure final product. Also, the possibility of obtaining the uranium residue in a concentrated form is an advantage. The use of but one chemical in making the extraction is advantageous. Taking the cost of alkali at $52 per ton and the maximum quantity required per ton of ore at 135 lb., the cost of this ingredient would be about $3.50 per ton of ore.

The cost of producing a pound of iron vanadate by this process is hard to give, as it was possible for me to handle charges of only 10 lb. at a time; but I believe that the cost will be lower than that of any other chemical process. An important matter at the present time is to create new uses for both vanadium and uranium. As long as the market for these metals remains limited it will be difficult to interest capital in this field.

Golden, Colo.

New Mining Laboratory of the University of Illinois.

The Department of Mining Engineering at the University of Illinois is just completing a new mining laboratory 100 ft. long by 42 ft. wide, divided into two equal sections, one for the treatment of ores and one for the washing of coal. The crusher room contains a Jeffrey coal roll, a pulverizer, an Allis-Chalmers gyratory crusher and a Colorado Iron Works 12 x 12 roll for ore. This end of the building also includes sample crushers and screens and two steel sampling floors, one for ore and one for coal.

The crushed material is elevated in a 15 x 20 continuous Peck carrier equipped to dump automatically along the upper run into any one of a row of steel bins, each holding five tons of coal. Beneath these bins is an Avery automatic traveling scale through which the material is delivered to any one of the screens or the washing or concentrating appliances. On the coal side there is a Holmes Brothers shaking screen and a Webster revolving screen, each fitted to separate four sizes and each about 13 feet long. On the ore side is a vibrating screen. Beneath each of these screens is a set of bins from which the screened material is taken by wheelbarrows to a dormant scale and then delivered to the lower run of the Peck carrier which elevates it to the upper tier of bins. Through the automatic traveling scale the coal is delivered to a three-compartment American concentrator jig or to a Stewart jig. A Luhrig jig and a Jeffrey-Robinson cone washer will be added later. The washed coal is delivered into a 1300-gal. settling tank from which it is elevated by a Webster bucket elevator to an overhead bin and from thence carried by the Peck carrier delivered to a bin outside the building, from which it is carted away to the boiler plant. The coal side of the laboratory has a capacity of 5 ton per hour.

On the ore side the material may also be crushed in a stamp battery. The fine screened ore is mixed with water and delivered by a Traylor centrifugal sand pump to three-compartment classifier, the products from which go to a 6-ft. Williams & Chalmers vanner, an 11-ft. Traylor concentrating table and a buddle. These three machines deliver to four Callow tanks beneath which are drying tables.

In the adjoining room is a completely equipped chemical laboratory and assay room for carrying on such tests as are required in the concentration of ores and the washing of coal. The equipment also includes an assortment of hand jigs, Munro classifiers, a spiral separator, a small Jeffrey-Robinson tub and a small laboratory concentrating plant made by the General Engineering Company. The machinery is being erected by the Burr Company, of Champaign, Ill.

The Mining Department is also equipping a new blasting and explosives laboratory, a rock drilling and coal-cutting laboratory and is building a new rescue station for giving training with oxygen helmets and other rescue appliances.

The offices, drafting rooms, library and recitation rooms of the Mining Department are located in the new engineering building, which will be ready for occupancy Sept. 15. In this building there is also a completely equipped laboratory for the study of mine gases and safety lamps.

Beginning with the collegiate year the Mining Department of the University of Illinois will thus have a very complete equipment for teaching all branches of mining engineering.

Drying Air for Blast Furnaces.

Mr. **Bruce Walter,** dry-blast engineer of the Carnegie Steel Company, of Pittsburgh, Pa., presented an interesting paper on this subject at the meeting of the Engineers' Society of Western Pennsylvania, held on April 16, 1912. The full paper is published in the May issue of the *Proceedings* of the society.*

The author remarks that up until very recently all of the dry-blast plants which have been built have followed very closely the original installation, made in 1904, at the Isabella furnaces of the Carnegie Steel Company.

In this **original dry-blast installation** a chamber is employed containing coils of pipe through which a cold calcium chloride solution is circulated, passing from top to bottom of the coils, the air passing upward through the spaces between the pipes. As the air rises the heat which it contains is transferred to the brine, and as its temperature is lowered its capacity for carrying moisture is reduced and the precipitated water is deposited on the pipes in the form of snow or frost. Two vertical sections of one of these chambers are shown in Fig. 1.

It will be seen that the pipe coils are in four separate compartments, any one of which can be thrown out of service without interference with the operation of the other three. This arrangement enables the operator to remove the frost without material interference with the efficiency of the plant as a whole. The defrosting must be done about every four days; that is, one compartment each day, and is accomplished by means of water sprays located at the top of the compartment. While the frost is being washed off, the air is shut out of the compartment and the brine is removed from the coil. The operation requires two or three hours.

There is no question as to the effectiveness of this method of drying air, as the moisture can be reduced to any desired point, depending on the temperature of the brine used and the amount of pipe surface to which the air is exposed, but there are certain disadvantages which are inherent in such installations and which are avoided in the latest designed plants. The most important disadvantage is the expensive construction required. The building itself is of steel framework construction well braced, and is sufficiently strong to withstand a sharply pulsating pressure; the walls are of first-class brickwork and the entire building is lined with a good insulating material which will withstand moisture and which also makes the building practically airtight. Much space is provided to give access to all parts of the chamber and the percentage of effective area is about 55 per cent of the ground space occupied.

Among the minor disadvantages in this original method are:

First—The freezing of the greater part of the moisture, a considerable saving in power being effected by precipitation without freezing.

* In connection with the subject of methods for drying air see the detailed description of the dry-blast plant of the Illinois Steel Co., South Works in our Vol. VI, p. 513 (Dec., 1908), and the Iron and Steel Institute paper, by F. A. Daubine and E. V. Roy on the desiccation of air by calcium chloride in our Vol. IX, p. 343 (July, 1911).

Second—The necessity, in order to avoid excessive expenditure for pipe surface, of maintaining the temperature of the brine 15 deg. lower than the temperature of air desired, thus increasing the cost of operating the refrigerating machinery.

Third—The expense of defrosting and the slight irregularities in the drying action due to this cause.

The foregoing disadvantages seem to be sufficient warrant for the dry-blast engineer to investigate the problem with the idea of decreasing the cost both of installation and operation and to secure, if possible, a more uniformly dry air.

Proposed New Method of Drying Air: One of the proposed methods of improving the operation of drying the air is shown in Fig. 2. This refrigerating chamber or drying tower consists of a cylindrical steel shell about 100 ft. high and 20 ft. in diameter, enclosing a smaller inner shell, the annular space between the shells being utilized for the purpose of removing the heat from the air. The air enters the tower at a pressure of about 2 ounces per square inch through pipe *A*, passes upward through pipes *B* and the hooded pipes *C*, where it comes into contact with mechanically cooled water dripping over the screens *D*.

The construction of these screens with the water distributing troughs *E* is shown in detail in section 7. The screens occupy the entire annular space and are arranged in radial sections, as shown in section 4. The water is pumped into the tower through pipe *F* and to tank *G*, from which it flows by gravity through regulating valve *H* to annular trough *I*, and thence by pipes *J* to pan *K*. From pan *K* it overflows to distributing troughs *E* through pipes *L*, which are so adjusted as to give an equal distribution of water over the entire annular space. After the water has passed over the screens it is collected in pan *M*, from which it flows through pipes *N* into filter *O*, where the dirt which is collected from the air is removed. The water passes from the filter through pipe *P* to the water coolers, where the heat is removed by means of refrigeration, after which it is again pumped to tank *G*.

When it is desired to clean the filter bed the valves in pipe *N* are closed, the water is taken to the coolers through pipe *O* and the filter bed is washed through pipe *P*, the overflow water passing out through trough *R* and pipe *S* to the sewer. As the air passes upward it is compelled to pass through the wetted screens many times and when it reaches the top of the grillwork it will be almost as cold as the entering water. In the design shown, if the entering water is 34 deg. Fahr. and the entering air 90 deg. Fahr., the temperature of the air will be reduced to 39 deg. and the temperature of the water will be raised to about 50 deg. The moisture content of the air will be 2.75 gr. per cubic foot after the air passes the eliminators *T*, which are intended to remove any entrained or unvaporized water.

In a similar manner the air passes through a second stage of cooling, passing through pipes C^1 and screens D^1, where it comes into contact with a non-freezing liquid, usually a solution of chloride of calcium, commonly called brine. The brine may be cooled to any desired temperature and is introduced into the system through tank G^1, from whence it is distributed over the grillwork, which is an exact duplicate of the first or water stage of cooling, with the exception that no filter is provided for the brine, as the air has been thoroughly washed before reaching the second stage. If it is desired to have a final moisture content in the air of 1/5 gr. per cubic foot, its temperature will be 24 deg., and the initial temperature of the brine must be 19 deg. The air then passes through pipe *U* to the blowing engines. The structure is entered through air lock *X*, and openings having a removable cover are provided in the inner shell at all points where it is desired to have access to the working parts of the tower. A general arrangement of a blast furnace equipped with dry blast, Fig. 3, shows the tower described in its relation to the refrigerating machinery, the blowing engines, the stoves, and the furnaces.

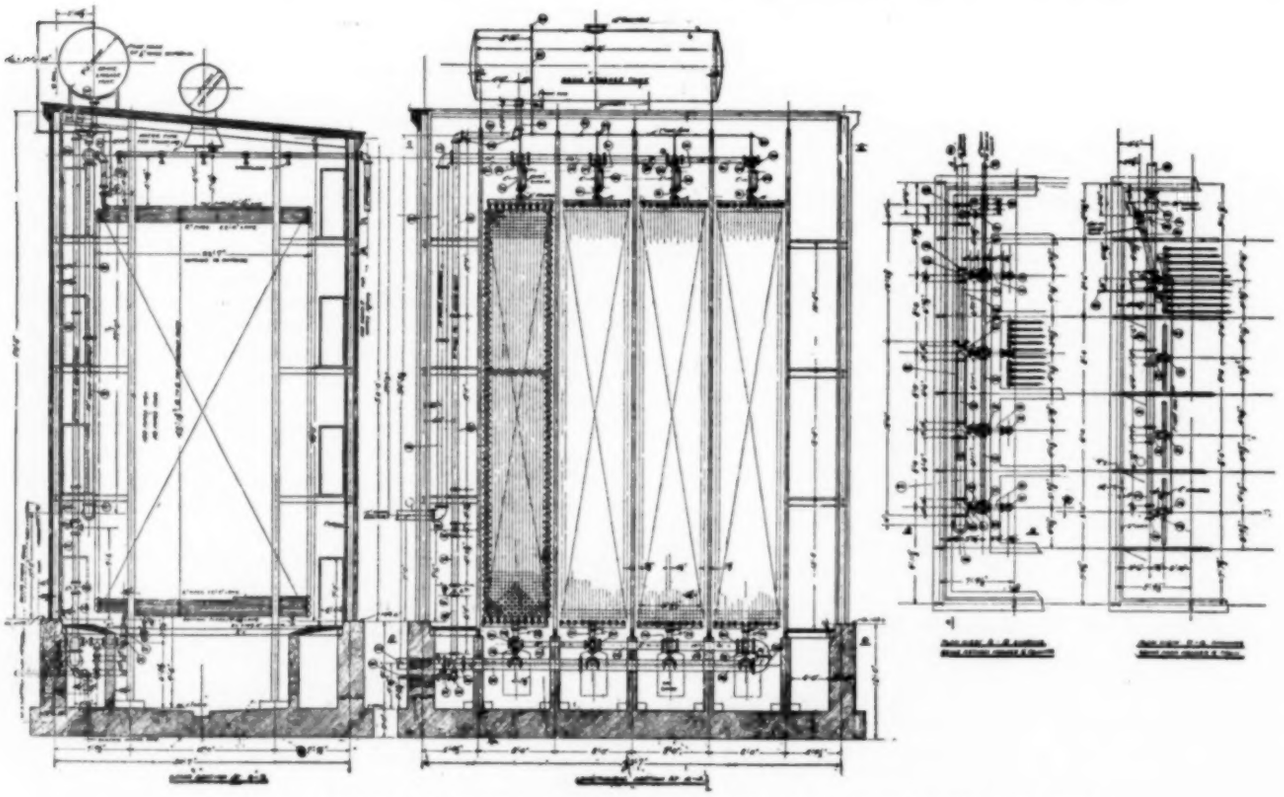


FIG. 1.—CROSS-SECTION OF AIR DRYING CHAMBER FOR COOLING AIR BY CONTACT WITH BRINE COILS.

Calculation of Heat to be Removed: In order to properly design an air-drying plant of any type it is first necessary to know the quantity of heat to be removed from the air. In order to show in a concrete manner how this may be done

Mr. Walter assumes a case of a furnace requiring 40,000 cu. ft. of air per minute, the initial temperature of which is 90 deg. Fahr. and the moisture content of which is 8 gr. per cubic

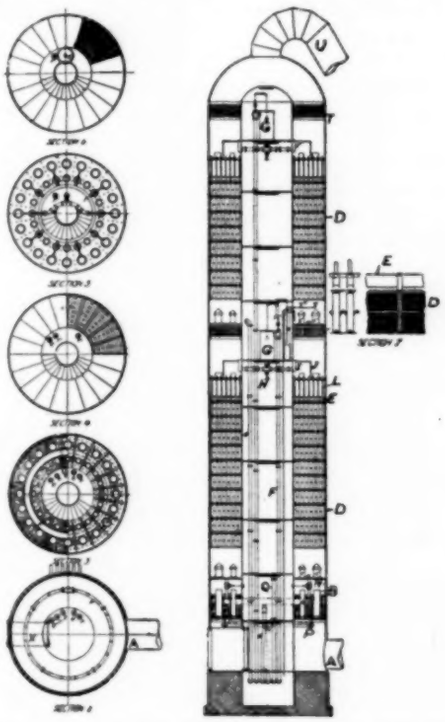


FIG. 2.—AIR DRYING TOWER FOR DRYING AIR BY THE TWO-STAGE, DIRECT CONTACT METHOD.

foot. It is desired to dry the air to such a degree that it will contain only 1.5 gr. of moisture per cubic foot, and as this corresponds to a saturated condition of air at the temperature of 24 deg. it is necessary to reduce the temperature from 90 deg. to this point. In the process described above this is done in two stages, the first, or water stage, reducing the temperature to 39 deg. with a moisture content of 2.75 gr. per cubic foot. and the second, or brine stage, in which the reduction is completed.

The heat to be removed in the first stage is determined as follows: 40,000 cu. ft. of air at 90 deg., containing 8 gr. of moisture, weighs 2867.4 lb., of which 45.7 lb. is water vapor and 2821.7 lb. is dry air. The heat to be removed from the dry air is, therefore,

$$2821.7 \times 0.2375 \times (90 - 39) = 34,178 \text{ B.t.u.}$$

The quanity of water vapor condensed and the heat rejected in the process is as follows:

The volume of dry air in the mixture at 39 deg. is

$$2821.7 \div 0.079843 = 35,340 \text{ cu. ft.}$$

In a saturated condition the volume is increased 0.8 per cent, or $35,340 \times 1.008 = 35,623$ cu. ft., which is the volume to be considered in the determination of the water vapor remaining in the air at this temperature. As saturated air at 39 deg. contains 2.75 gr. of moisture per cubic foot, the quanity of water vapor remaining is

$$35,623 \times 2.75 \div 7000 = 14 \text{ lb.}$$

As the original quantity was 45.7 lb., the quantity condensed by the reduction of temperature is 31.7 lb. There is no condensation until the temperature of 70 deg. is reached, as the air is not in a saturated condition above this point; each degree of reduction after passing the 70 deg. mark produced a condensation of water vapor, but the quantity condensed becomes less as the temperature is lowered; thus from 70 to 69 deg. the quantity of water vapor is reduced 0.25 gr. per cubic foot, and from 40 to 39 deg. the reduction is 0.1 gr. One-half of the 31.7 lb. is condensed when a temperature of 59 deg. is reached, and it is assumed that the latent heat of condensation for this temperature, which is 1058.4 B.t.u., is practically correct for determination of the work of condensation of all the water vapor precipitated, and the heat to be removed is

$$31.7 \times 1058.4 = 33,551 \text{ B.t.u.}$$

The remainder of the work of heat removal consists of cooling the water vapor to the condensation point, cooling the water vapor not condensed, and cooling the condensed water to the temperature of the cooling water leaving the refrigerating chamber. As the mean temperature of condensation is 59 deg., we may consider that all the water vapor is cooled from 90 to 59 deg., and the heat to be removed, its specific heat being 0.475, is

$$45.7 \times 0.475 \times (90 - 59) = 673 \text{ B.t.u.}$$

The water vapor not condensed is cooled from 59 to 39 deg. and the heat to be removed is

$$14 \times 0.475 \times (59 - 30) = 133 \text{ B.t.u.}$$

Assuming that the water leaves the refrigerating chamber at 50 deg., the condensed water vapor must be cooled from 59 to 50 deg., and the heat to be removed is

$$31.7 \times 9 = 285 \text{ B.t.u.}$$

The total heat to be removed in the first stage is, therefore, 68,820 B.t.u. Assuming that the initial temperature of the cooling water is 34 deg. and its final temperature 50 deg., there must be circulated $68,820 \div (50 - 34) = 4301.25$ lb. of water, equal to 516 gal. per min. To this must be added the heat absorbed from the atmosphere, which in any well-insulated plant will not be more than 1 per cent. This makes the total heat to be removed by the refrigerating plant 69,508 B.t.u. per min. And the water to be circulated 521 gal. per min.

In the second stage the brine enters at a temperature of 19 deg. and leaves at 29 deg. The air enters at 39 deg. and leaves at 24 deg. The heat to be removed is as follows:

Heat to be removed from dry air is

$$2821.7 \times 0.2375 \times (39 - 24) = 10,052 \text{ B.t.u.}$$

Volume of dry air at 24 deg. is

$$2821.7 \div 0.0823 = 34,286 \text{ cu. ft.}$$

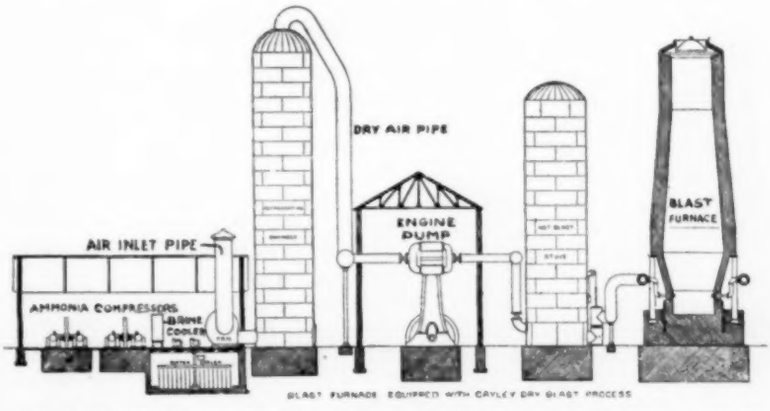


FIG. 3.—DIAGRAMMATIC SKETCH OF BLAST FURNACE EQUIPPED WITH GAYLEY DRY BLAST PLANT.

Volume when saturated is

$$24,286 \times 1.0043 = 34,433 \text{ cu. ft.}$$

Quantity of water vapor remaining in mixture is

$$34,433 \times 1.5 \div 7000 = 7.3785 \text{ lb.}$$

The water vapor condensed is

$$14 - 7.3785 = 6.6215 \text{ lb.}$$

Heat to be removed for condensing water vapor is

$$6.6215 \times 1073.4 = 7108 \text{ B.t.u.}$$

in which 1073.4 is the latent heat at 32 deg., which is the mean temperature of condensation.

Heat to be removed from water vapor before condensation is

$$6.6215 \times 0.475 \times (39 - 32) = 22 \text{ B.t.u.}$$

Heat to be removed from water vapor not condensed is

$$7.3785 \times 0.475 \times (39 - 24) = 53 \text{ B.t.u.}$$

Heat to be removed from condensed water vapor is

$$6.6215 \times (32 - 29) = 20 \text{ B.t.u.}$$

The total heat to be removed in the second stage is, therefore, 17,255 B.t.u. Adding 1 per cent for heat leakage through the insulation, the quantity becomes 17,428 B.t.u. Assuming that the brine weighs 10 lb. per gallon and has a specific heat of 0.7, the quantity to be circulated, when heated by the air from 19 to 29 deg., is

$$17{,}428 \div [10 \times 0.7 \times (29 - 19)] = 294 \text{ gal. per min.}$$

New Method for Concentrating Brine: It is obvious that some means must be provided for removing the accumulation of water from the brine used in the second stage of cooling. The quantity of water added is 6.6215 lb. per min., and it is desirable to evaporate this with as little disturbance to the regular action of the plant as possible. An automatic device is shown in Fig. 4.

A sample of brine is continuously taken from the system by means of several funnels *V*, Fig. 2, Section 1, into tank *W*, and

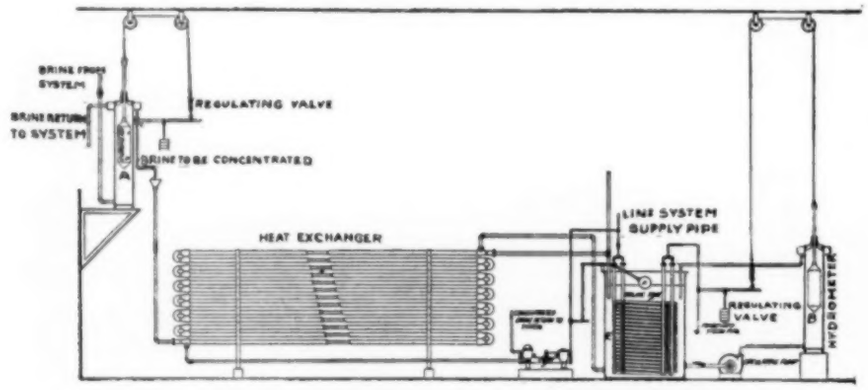


FIG. 4.—AUTOMATIC BRINE CONCENTRATOR.

overflows into pan M^1. This constant level tank is tank *A* in Fig. 4.

A hydrometer *B* floats in the tank and operates valve *C*, which is adjusted to open when the brine falls below a desired specific gravity. This brine, which is drawn off at a temperature of 29 deg., flows through a heat exchanger *D* to boiler tank *E*, where the volume of brine is reduced one-third by boiling. The heat is furnished by a live steam coil *F* and is regulated by the density of the brine. A pump circulates a small portion of the liquid through the constant level tank *G*, where the hydrometer *H* acts upon the exhaust from the steam coil, supplying more or less heat as the density of the brine decreases or increases. The pump *J* returns the brine through the heat exchanger to the brine coolers. This pump is governed by float *K* in the boiling tank.

If it is desired to have the brine in the system at 1.2 specific gravity it is necessary to draw off by means of this apparatus about 3 gal. per minute and boil it down to 2 gal. By installing an efficient heat exchanger the heat returned to the system is reduced to a negligible quantity. Any convenient source of heat which will give a temperature of 250 deg. can be used for boiling the brine, and in many cases gas will be found less expensive than live steam.

Refrigerating Machinery: Having determined the heat to be removed from the air in each stage, it is necessary to install the proper refrigerating machinery to remove these same quantities of heat from the water and brine. A specification can be drawn up from the foregoing figures giving the quantities of water and brine, the temperature range through which they are to be cooled, the steam pressure available, and the temperature of the water supply for the ammonia condenser. Having this information, any builder of refrigerating machinery can quote a price for an installation guaranteed to do the required duty.

Power Required to Operate the Plant: The power required to operate the plant varies with atmospheric conditions, and Table No. 1 shows the maximum and average requirements. The saving of power in the blowing engines is due to the reduction of volume in the air blown, and is based on average temperature and humidity conditions in the blowing engine room. The blast pressure is assumed to be 15 lb. per square inch.

TABLE NO. 1.

Power required to operate a two-stage Gayley dry-blast plant having a capacity of 40,000 cu. ft. of air per min. at a temperature of 90 deg. Fahr. with a moisture content of 8 grains per cu. ft.

	Maximum	Average
Ammonia compressors, first stage	433 i.h.p.	152 i.h.p.
Ammonia compressors, second stage	137 i.h.p.	96 i.h.p.
Total	570 i.h.p.	248 i.h.p.
Steam for brine concentrator, lb. per hour	544	409
Motor-driven fan	60 e.h.p.	60 e.h.p.
Motor-driven water pump	19 e.h.p.	14 e.h.p.
Motor-driven brine pump	11 e.h.p.	11 e.h.p.
	90 e.h.p.	85 e.h.p.
Average saving of power in blowing engines		270 i.h.p.

Cost of Operation: The cost of operating the plant described will vary to some extent with local conditions. In the Pittsburgh district, at the present rate of wages, the cost will amount to $6,800 per year, which is equivalent to 4¼ cents per ton of iron produced for a furnace making 160,000 tons per year. This estimate includes labor, power, water, repairs, oil, waste, packing, ammonia, and chloride of calcium. A plant of double the size of the above can be operated at $11,000 per year, making the cost per ton of iron 3½ cents.

Detailed Cost of Installation: The cost of installation is estimated as follows:

Building	$9,400
Refrigerating chamber	23,300
Refrigerating machinery	34,300
Pumps and motors	1,400
Fan and motor	3,500
Brine concentrator	800
Piping and covering	3,400
Electric wiring for power and light	1,000
Steam condenser	1,300
Engineering and incidentals	7,700
Total	$84,800

This estimate does not include steam and water pipe lines to the plant, or air piping from the plant to the blowing engines, as these items depend very largely upon local conditions. Under moderately favorable conditions these items should not cost more than $8,000, making the total cost of installation $92,800. A plant having a capacity of 80,000 cu ft. of air per minute requires the expenditure of $170,000. These figures are 25 per cent below the lowest installation costs of plants in which the air cooling is done by the brine coil system.

This expenditure is compared by Mr. Walter with the savings attained in blast furnace operations, as follows: It is conservatively estimated by those who have operated dry-blast furnaces that a saving in fuel of 12.5 per cent is effected, and, at the same time, the production of iron is increased 10 per cent. This means that if a furnace uses 2400 lb. of coke to make 1 ton of iron and has a production of 400 tons per day,

the fuel saved by dry blast is 300 lb. per ton and the production is increased 40 tons, all of which is accomplished without increased labor cost. The furnace operation can be varied to suit the market conditions so that, if it is desired, a larger increase in production can be made while saving a smaller percentage of fuel, or a larger saving of fuel can be effected with a corresponding decrease in production. Another important advantage is that the furnace receives a uniform weight of air with a uniform moisture content, and, therefore, works with greater regularity, producing a more uniform product than one blown with natural air. The necessity of carrying reserve heat in the hot blast stoves is also eliminated, as the irregularities which necessitate this precaution do not occur. By reducing the fuel the limestone required is decreased. Regularity in working means a reduction of flue dust carried off by the gases. The actual profit in dollars and cents from the installation of a dry-blast plant is a separate problem for each blast furnace. A fair figure for the Pittsburgh district is a net saving of 50 cents per ton of iron, which, for a furnace making 160,000 tons per year, is $80,000 annually.

* * *

Mr. Walter's paper elicited quite an extended discussion.

Mr. **W. H. Carrier,** of the Carrier Air Conditioning Company, pointed out that the latest and best data on water vapor are to be found in Marks and Davis' steam tables and in W. H. Carrier's paper on rational psychrometric formulæ in the December, 1911, issue of the *Proceedings* of the American Society of Mechanical Engineers. In the latter paper a simple method is described for obtaining the refrigerating requirements in cooling and dehumidifying air. The cardinal principle of this method is that for a given temperature of evaporation; that is, for a given wet bulb temperature, a pound of air contains a fixed and definite amount of heat, regardless of the dry bulb temperature or of the moisture contents. This means that the wet bulb temperature is the rational and convenient measure of the total heat contents of the air, both latent and sensible.

For example, an ordinary recording thermometer with the bulb covered with a wetted wick, over which a current of air is circulated, could be made to record B.t.u. contents per pound of air by graduating the record in terms of B.t.u. instead of in degrees temperature. Consequently in dehumidifying the maximum refrigerating load will occur at the maximum wet bulb temperature, relation of dry bulb temperature to moisture contents being immaterial.

The condition which Mr. Walter assumes is 40,000 cu. ft. at 90 deg. and containing 8 gr. of moisture. This is equivalent to saying that the wet bulb temperature is 76.2 deg. This, however, to Mr. Carrier's knowledge does not represent the extreme conditions. He has personally observed in Buffalo wet bulb temperatures as high as 78 deg., and thinks possibly they may be found in Pittsburgh as high as 79 or 80 deg., which would considerably increase the refrigerating requirements.

Mr. Carrier also thought that the calculations could be much simplified by basing them on pounds of air instead of cubic feet of air. The calculation may be made as follows: We will make the assumption of Mr. Walter, which is 2821.7 lb. of dry air, containing 45.7 lb. of moisture at a temperature of 90 deg. This will then give us $45.7 \div 2821.7 = 0.00162$ lb. of dry air, or $0.00162 \times 7000 = 113.4$ gr. of moisture per pound of dry air.

The volume of 1 lb. of dry air plus 113.4 gr. moisture, we find from the chart, is 14.22 cu. ft. This will give us $28\,217 \times 14.22 = 40{,}120$ cu. ft., instead of 40,000 cu. ft., given by Mr. Walter.

From the writer's psychrometric chart we find air at 90 deg. and containing 113.4 gr. per pound of dry air has a temperature of evaporation, or true wet bulb temperature, of 76.2 deg. Taking a point on the total heat curve corresponding to this temperature, we find that 1 lb. of air contains 38.85 B.t.u. Assuming this cooled to a saturation temperature of 39 deg., we find the corresponding heat content to be 14.9 B.t.u. per pound of dry air, requiring a removal of 23.95 B.t.u. per pound of dry air. We will, therefore, have a total refrigeration requirement per minute of

$$2821.7 \times 23.95 = 67{,}670 \text{ B.t.u.}$$

This, however, does not include the heat of the liquid, which has to be cooled from a wet bulb temperature of 76.2 deg. to 39 deg., the final water temperature. As there are 45.7 lb. of water in the air, 1700 B.t.u. will be required for cooling this, giving us a total refrigerating requirement of 69,310 B.t.u. per minute, or approximately 350 tons refrigeration. Allowing for the rise of the water temperature to 50 deg., there will be a saving of 355 B.t.u., reducing the total requirement to 69,015 B.t.u. per minute.

In the second stage, as before, we have a saturation temperature of 39 deg. at which the air contains 14.9 B.t.u. per lb. of dry air. This is then cooled to a saturation temperature of 24 deg., at which, according to the psychrometric chart, it contains 8.55 B.t.u. per pound of dry air, making a removal of 5.35 B.t.u. per pound of dry air. The total B.t.u. removed per minute is then $2821.7 \times 5.35 = 15{,}095$ B.t.u. The refrigeration required in reducing the heat of the liquid is for 13.7 lb. through a range of 15 deg., or 200 B.t.u. per minute. This gives a total of 15,295 B.t.u.

Mr. Carrier finally mentioned that a dehumidifying plant embodying many of the principles contained in Mr. Walter's proposed new method of drying air was installed under the writer's supervision by the Carrier Air Conditioning Company and successfully operated in 1908. This plant is in operation to-day, its capacity having been doubled by the purchase of an additional unit, the exact duplicate of the first.

In the second stage the essential principle involved is the direct contact of the brine with the air to be dried. The practicability of this process was demonstrated by Mr. Carrier in Mr. Walter's presence in June, 1909, using an experimental apparatus having a capacity of 10,000 cu. ft. per minute in which the air was cooled as low as 20 deg. with an inappreciable entrainment of calcium chloride as determined by careful chemical analysis. The chief difference between Mr. Carrier's process and Mr. Walter's process appears to be in the fact that Mr. Carrier introduces the refrigerating liquid directly into the air in the form of a finely divided spray, which is afterwards eliminated, while Mr. Walter distributes his brine over screens.

To Mr. Carrier's mind the latter process has several mechanical objections, the chief of these being the difficulty of distributing water uniformly over the screens and secondly the rapid corrosion of the screens which must necessarily take place in an atmosphere quite noticeably charged with sulphur dioxide or traces of sulphuric acid as in the Pittsburg district.

Mr. Carrier has found in these plants that the recirculated water becomes highly acid in a short time and will corrode practically everything but monel metal and cast iron. The pumping equipment and water piping must, therefore, be carefully considered with reference to its acid-resisting construction. Mr. Carrier would recommend extra heavy piping, monel metal nozzle and eliminators. He also recommends so far as possible the elimination of all easily corrosible parts, such as screens, etc., unless they are constructed of monel metal or other acid-resisting material.

Mr. **Louis Block,** of New York City, was the next speaker. He mentioned the fact that if air is blown over or through a calcium chloride solution, particles of calcium chloride are carried along with the air and are deposited upon the surfaces with which the air afterwards comes in contact. In a certain fur storage plant the air was blown over a cooled calcium solution and the method of cooling had to be abandoned because the chloride was distributed to a considerable extent over the fur garments hanging in the cooling room. The question arises, will the calcium chloride injuriously affect the inner surfaces of the blowing cylinders? Will it affect the product of the blast furnace?

Mr. Block further thought that if regularity of blast-furnace operation is the all-important consideration, it will be sufficient to remove the moisture out of the air down to 2.75 (39° F.) or 2.50 gr.—which is a perfectly simple proposition—and not to attempt to remove another 1.25 gr. of moisture down to 1.5 gr. (240° F). This second step is cumbersome and expensive and Mr. Block gives figures to show that this second step probably costs more than it saves, and introduces the liability of irregularity in blast-furnace operation.

Mr. **J. B. Ladd**, of Philadelphia, asked on what calculations or actual results the author's statement was based that the saving in blowing engine power is practically equal to the power required for operating the refrigerating plant, so that the power for refrigeration is obtained at no expense.

Mr. **S. M. Marshall**, of the Cambria Steel Company, of Johnstown, Pa., pointed out that there are three methods of refrigeration which are being considered more or less to-day; one cooling the air before the blowing engine, one after the blowing engine, and the dense air system by which the air itself is compressed under the ordinary blast pressure, cooled at this higher pressure, and later re-expended to reduce the temperature and moisture. He asked Mr. Walter as to the merits of his system as compared with the other two. Further the percentage of saving due to the use of dry blast depends on the initial coke consumption of the furnace and upon the stove heats. How do these two factors affect the percentage of saving? Moreover, dry blast and thin lining in blast furnaces produce apparently large savings in the fuel economy when used independently. Are there any data on the probable effect of using the two simultaneously?

Prof. **S. L. Goodale**, of the University of Pittsburgh, emphasized that a saving of 20 per cent of fuel alone in such a big fuel-consumer as a blast furnace, is itself a very important item. The statement of Mr. Walter that one pound of coke is required to break up one pound of water vapor, he considers to be slightly inaccurate. In the reaction for breaking up the moisture we may say that 12 parts of carbon are needed, actually, to break up 18 parts of water vapor; and there is an absorption of heat in that reaction which must be made up by the combustion of an additional 12 parts of carbon. So that if 24 parts of carbon were consumed to 18 parts of water vapor broken up the heat produced would be practically balanced by the heat absorbed; and in addition a little carbon must be burned to heat the products of this reaction to the temperature of the furnace hearth. So that it seems that about 1½ pounds of coke is a more close approximation to the amount that is actually used by the furnace in breaking up one pound of water vapor.

As to the two-stage condensation, Prof. Goodale said that most of the work is done with the cheaper energy. That is, it is more expensive to remove a certain amount of water vapor at a very low temperature than to remove the same quantity at a temperature nearer the ordinary; and as Mr. Walter proposes to do most of the condensing at a temperature economically obtainable, compared to freezing and undercooling, a considerable economy of operation should result. At the opposite extreme of temperature, in other operations, vast quantities of heat are thrown away by metallurgical plants in waste furnace gases because heat is not usable by the particular furnace unless supplied at a high temperature. In some cases heat of this kind is recovered for steam raising; but Prof. Goodale believes that there is still a field from which good technical work can be made to yield large returns by utilizing the heat for purposes for which moderate temperatures only are rquired. This is just another problem in multi-stage operation for economy.

In regard to the amount of power used, it would seem as if there were a sufficient amount of furnace gas available around blast furnaces that could be used, and there should not be needed any great additional expense for the power used in the refrigerating plant.

There is another method of drying blast which has been introduced at one place abroad, and that is direct drying by contact with dry calcium chloride. Prof. Goodale asked if it would be possible perhaps to substitute that method of drying in place of the proposed second stage. In that way would be secured the advantage of the uniform moderate temperature of the blast, and possibly a lower cost for the final drying of the air.

Mr. **E. D. Leland**, superintendent of the compressing stations of the Philadelphia Company at Pittsburg, inquired concerning another plan of drying air proposed as cheap and efficient. The idea is to spray glycerine into the air cylinder of a blast engine, in order that the glycerine might absorb the moisture and thus give dry air for the furnace. A separating receiver was to be introduced between the engine and the furnace, and the glycerine thus recovered was to be used again. Several years ago the device was patented, and an experimental plant erected at Bradford, Pa., from which very satisfactory results were claimed.

In Mr. Walter's sketch, after the second refrigeration, the air goes directly from the top of the tank to the blast engine. Now, after cooling, the excess of moisture would doubtless be present in the air in the form of fog, and this fog might continue on through with the air current into the furnace. This action is probably what did occur in the case mentioned where chloride of calcium spray was carried over to the injury of some furs. It requires time to precipitate a light fog, and Mr. Leland thought that by passing the air through a large tank or drum, after its final cooling, the temporary checking of the air current would give time for moisture precipitation and thus add materially to the efficiency of the new apparatus.

Mr. **J. J. Lilie**, of the Carrier Air Conditioning Company, of New York City, asked as to the kind of insulation employed by Mr. Walter. He also gave figures on some power requirements not considered by Mr. Walter. These are the frictions of the air going through the cooling chamber and the work lost by the falling of the water and the falling of the brine, the water and brine being pumped to tanks and allowed to flow down. These losses he found to amount to 2.3 per cent.

Mr. Lilie then replied to the criticism which had been raised that there was liability of entrainment and carrying over of a certain amount of calcium chloride. In installations using a direct spray, which may be found in other industrial work, if a proper velocity is kept through the eliminating plates and they are properly constructed, there is no entrainment passing the eliminating plates such as Mr. Walter proposes to use and a chamber such as Mr. Leland had mentioned is not necessary. Mr. Lilie had recently seen the records of some very accurate tests worked out by people "who were in no way interested in the plant which showed as little as 0.07 of a grain of moisture as the maximum amount carried over in a test extending over a period of several days. This was the maximum reading they found in taking the temperature of the dew point and the temperatures after they had been heated up, that is, taking a sample from the bustle pipe."

Mr. Lilie finally pointed out that Mr. Walter's system, as it is, would not deliver to the furnace a constant weight of air, on account of the variations of the barometric pressure. A variation in barometric pressure affects the amount of air delivered to the furnace because the weight is inversely in proportion to the pressure, and if we have in one day a variation in barometric pressure of 1½ in. it makes a variation in the amount of air delivered to the furnace of over 5 per cent. Hence, by maintaining a constant speed of our engine and delivering to the blowing engine air at a constant temperature, we are not necessarily delivering a constant weight of air. If it is of importance to deliver a constant weight of air we ought to take the barometer into account as well as the temperature.

In his final reply the author, Mr. **Bruce Walter**, first answered Mr. Ladd's question as to the power saved in the

blowing engines, which is simply a matter of volume of air blown.

A fair estimate of the power required to compress air in a blowing engine to 15 lb. gauge pressure, is 60 i.h.p. per 1000 cu. ft. of air per minute so compresesd. Assuming that the air is taken from the engine room and that the average temperature is 85 deg. and the average moisture content is 4.8 gr. per cubic foot, then the power required to compress 1000 cu. ft. of this mixture is 60 i.h.p. and the weight of dry air in mixture is 71.96 lb. The power required to compress one pound of dry air under these conditions is 0.8338 i.h.p.

When the air is dried before compression to 1.5 gr. water vapor per cubic foot, it is assumed that a rise of five degrees in temperature occurs between the drying tower and the blowing engines, so that a condition of 29 deg. and 1.5 gr. exists at the inlet valves and the weight of dry air in 1000 cu. ft. of this mixture is 81.13 lb.

The power required to blow one pound is

$$60 \div 81.13 = 0.7395 \text{ i.h.p.}$$

The power saved on the delivery of one pound of dry air is, therefore, 0.0943 i.h.p., equal to 11.3 per cent of the power required to blow one pound under the natural air condition.

In reply to Mr. Marshall, Mr. Walter said he was not prepared to discuss in detail the system of refrigeration after compression. He offered, however, the criticism that we lose something in regularity in quantity of oxygen delivered to the furnace by refrigerating after compression unless some effort is made to regulate the speed of the blowing engines to meet the temperature and moisture conditions in the blowing engine room.

We also lose from 80 to 100 deg. of heat by drying after compression. In drying before compression we get the benefit of the heat of comparison added to the initial temperature of the air entering the blowing engines, but in the after-compression method, all the heat of compression is removed and the temperature is further lowered to 40 or 50 deg. This lost heat must be supplied by the hot blast stoves and, therefore, the gas consumed in furnishing it should be charged against the plant as power consumed.

As to the dense air drying, Mr. Walter referred to some tests made some years ago where the cost was found to be excessive. It is nothing but an air refrigerating machine and air refrigeration is much more costly than ammonia refrigeration for the B.t.u. removed.

Mr. Walter's insulation figures are based on covering the tower with two inches of cork lagging from the bottom of the filter to the water distributing pan and three inches of the same material from this point to the air outlet. The exact calculations show a loss of 504 B.t.u. per minute, which is less than 0.6 per cent of 86,075 B.t.u., the whole work to be done on the air.

In regard to the barometric pressure that is a matter over which we have no control, and Mr. Walter thinks that our efforts at regularity in temperature and moisture are commendable even if we do have some irregularity of barometric pressure. In other words, we should not stop at efforts to deliver a regular quantity of oxygen to the furnace because of the barometric changes.

By introducing the air to the blowing engines at a constant temperature and moisture content, we eliminate the greatest variable and prevent a combustion of an abnormally low barometer with a high temperature.

In reply to Prof. Goodale Mr. Walter said that if it requires one and a half pounds of coke to break up one pound of water, the saving of fuel due to the elimination of moisture is a larger percentage of the total saving than he had assumed in the paper and the saving due to regularity is proportionally reduced. This being true, Mr. Block's calculation will, therefore, have to be revised and will then show a considerable saving in fuel for the second stage.

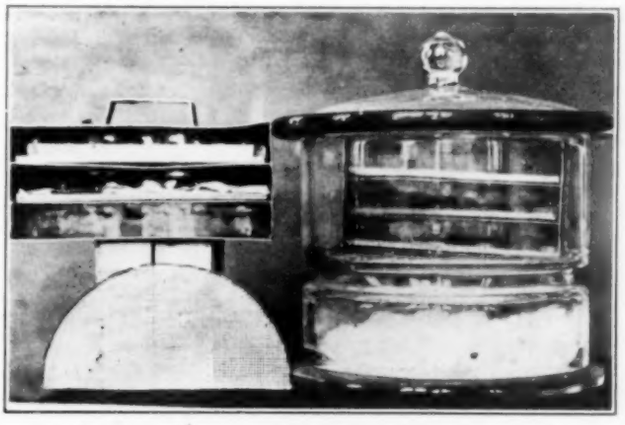
In regard to Prof. Goodale's suggestion of using calcium chloride in a crystallized form for removing the moisture in the second state, Mr. Walter said that this method of drying the blast was tried at a furnace in France. The towers were provided to be used in a regenerative manner, so that while the air was being dried in the first tower, in the second the calcium chloride was being dried by heat from waste gases, and in the third the calcium chloride was being cooled to be ready to dry air again. But it was found impossible to keep the calcium chloride from fusing into a solid mass which was not broken by the method used in drying it and consequently had to be broken up by hand. While the air was very dry when the calcium chloride was fresh, the moisture rose rapidly and the tower had to be changed very frequently.

In reply to Mr. Carrier, Mr. Walter thought that he overestimates the difficulties to be encountered due to the corrosion of the screens and surfaces exposed to the air and the circulating water. While this water in time may become acid, it can easily be changed and a new charge of purified water put into the system. In fact this will be done automatically to a large extent when the water wasted from washing the filter is replaced by new water. If this is found to be insufficient an occasional introduction of lime will be found efficient in preventing the corrosive action of the acid.

Metallography Sample Holder.

BY STEPHEN L. GOODALE

The accompanying photograph shows a holding device planned by the writer for use in the Metallography laboratory of the University of Pittsburgh. Its utility lies in storing a considerable number of polished small samples of iron and steel in a desiccator in such a way that any sample is easily



METALLOGRAPHY SAMPLE HOLDER.

accessible without disturbing the arrangement of the rest. None of the preservative coatings that we have tried are satisfactory, as the view through the microscope is invariably somewhat blurred by the thin film of varnish.

This holder is made of ordinary tin in two parts, each of a half circle plan and several inches high, the two together just filling an eight inch desiccator. The shelves are made of perforated tin, and on each shelf is placed a perforated loose pan of the same shape and size as the shelf itself, and with its edges slightly turned up. A soft paper or cloth cover is used over the tin pan in order not to mar the fine polish of the samples.

For any special series of samples, as for instance a set to show varying heat treatment on any steel, we use a piece of quarter or half-inch wood cut to the size of the pan. Depressions are cut in the wood with a twist drill and numbered, so that it looks like a miniature biscuit pan when finished, each sample in the series having its own place.

It is well to use the porcelain rest in the desiccator under

this holder, as the sloping glass walls of the desiccator do not hold one half of it securely if the other is taken out for examination. In that case the half left in the desiccator may slip down and mix up the samples and also rub their polished surfaces.

We have found it advisable to have a large number of prepared samples ready for our students to examine, with the record of just what each one is and its heat treatment. Each student is given a sufficient number of pieces to polish and etch so that he becomes thoroughly familiar with those operations, without having to devote so much time to the purely mechanical part of the work as to be objectionable. Besides this each man is required to repolish and re-etch a number of the prepared samples, and study the different effects of varying the etching, etc., on one sample.

The photograph shows one half of the holder in place in the desiccator, the other half outside, with a wood tray on top shelf, a pan with samples on middle shelf, and an empty pan standing below. *University of Pittsburgh.*

Determination of Very High Temperatures

By G. A. Shook.

Radiation Laws of Planck and Wien.

In the earlier papers of this series, published in the April, June and July issues of this journal, the methods of calibrating pyrometers on the basis of Wien's law were taken up. Within the limits of experimental error, Wien's law will hold for all temperatures from 600 deg. to 3000 deg., and, of course, this includes all of the temperatures ordinarily met with.

It can be shown in several independent ways, however, that for large values of temperature and wave-length, in the visible spectrum, Planck's law represents more nearly the relation between the temperature T and the luminous intensity J for a particular wave-length λ.

It is now necessary to make a comparison of these two laws in order to determine what error will be committed by the use of Wien's law for temperatures of the magnitude of 6000 or 7000 deg. C., the probable temperature of the sun.

Planck's law may be written in the form

$$J(e^{c_2/(\lambda T)} - 1) = C_1\lambda^{-5} \qquad (1)$$

Wien's law may also be written

$$J = C_1\lambda^{-5}e^{-c_2/(\lambda T)} \qquad (2)$$

Obviously if we can neglect the -1 in Planck's formula, equation (1), it will reduce to Wien's formula. This can only be done when $e^{c_2/(\lambda T)}$ is large compared to unity.

Suppose we take λ as 0.6μ (red light); since $C_2 = 14{,}500$, if T is not too large, the quantity $e^{C_2/(\lambda T)}$ will be large in comparison to 1. We may calculate its value as follows:

Let

$$e^{C_2/(\lambda T)} = x$$

Taking the logarithm of both sides of this equation

$$C_2/(\lambda T) = \log x$$

or

$$\frac{\log_{10} e\, C_2}{\lambda T} = \log_{10} x = \frac{K_2}{T}$$

where $\log_{10} x$ means the *common logarithm of* x. $\text{Log}_{10} e = 0.4343$.

Therefore,

$$\frac{0.4343 \times 14{,}500}{0.6} = K_2 = 10{,}500.$$

Taking T as 1000 deg. (absolute Centigrade)

$$10{,}500 \div 1000 = 10.5 = \log x,$$

whence $x = 31{,}630{,}000{,}000$.

Taking a somewhat larger value of T, say $T = 3000$ deg., about the temperature of the carbon arc

$$10{,}500 \div 3000 = 3.5 = \log x$$

and

$$x = 3100.$$

For $T = 6000$ deg., the temperature of the sun

$$10{,}500 \div 6000 = 1.75$$

whence

$$x = 56.24.$$

Therefore,

$$e^{\frac{C_2}{\lambda T}} = 31{,}630{,}000{,}000 \text{ for } T = 1000$$
$$= 3163 \text{ for } T = 3000$$
$$= 56.24 \text{ for } T = 6000$$

Hence it is evident, as far as J is concerned, that for large values of T, the 1 in equation (1) cannot be neglected; *i. e.*, Wien's law is only an approximation to Planck's law for these high temperatures.

The effect of this error on the measurement of temperature will now be considered. Equation (1), Planck's law, may be written:

$$(e^{C_2/(\lambda T)} - 1)\, J = C_1\lambda^{-5} \text{ or}$$
$$Je^{C_2/(\lambda T)} = C_1\lambda^{-5} + J.$$

Replace $C_1\lambda^{-5}$ by K' and take the logarithm of both sides of this last equation.

Hence

$$\log Je^{C_2/(\lambda T)} = \log (K' + J)$$

The logarithm of a product is the sum of the logarithms of each quantity and $\log e = 1$, therefore

$$\log J + \frac{C_2}{\lambda T} = \log (K' + J) \text{ or}$$

$$\frac{C_2}{\lambda T} = \log \left(\frac{K' + J}{J} \right)$$

Whence

$$T = \frac{C_2}{\lambda \log (1 + K'/J)}$$

and

$$t = \frac{\log_{10} eC_2/\lambda}{\log_{10} (1 + K'/J)} - 273$$

Let

$$K_2 = \log_{10} e.C/\lambda$$

therefore

$$t = \frac{K_2}{\log_{10} (1 + K'/J)} - 273 \qquad (3)$$

Equation (2), Wien's formula, can also be written

$$J = C_1\lambda^{-5}e^{-C_2/(\lambda T)}$$

Taking the log of both sides of this equation,

$$\log J = \log_{10} C_1\lambda^{-5} - C_2/(\lambda T) \text{ or}$$
$$\log_{10} J = K_1 - K_2/T$$

where

$$K_2 = \frac{C_2 \log_{10} e}{\lambda}$$

and

$$K_1 = \log_{10} C_1\lambda^{-5} = \log_{10} K'.$$

Wien's law now becomes

$$t = \frac{K_2}{K_1 - \log J} - 273 \qquad (4)$$

The application of equation (4) was taken up fully in my article in the July issue. The expression "log" will henceforth be taken to mean the "common log." To compare equation (4) with equation (3) we may rearrange (4) as follows:

$$t = \frac{K_2}{\log K' - \log J} 273$$

or

$$t = \frac{K_2}{\log (K'J)} - 273 \qquad (5)$$

A comparison of equations (3) and (5) will show at once that when the 1 in Planck's formula, equation (3), is neglected the latter reduces to Wien's formula, equation (5).

Suppose we are using a Wanner pyrometer for determining high temperatures. To simplify the calculations we will assume convenient but plausible values of the various quantities in the above equations. In order to determine K' assume that $\phi = 45$ deg. when $T = 1300$ deg., then $J = \tan^2 45 = 1$, *i. e.*,

at this particular angle the temperature of the standard source must equal the temperature of the body under consideration.

Substituting these values in equation (3)

$$1300 = 10{,}500 \div \log (K' + 1)$$

and

$$\log (K' + 1) = 10{,}500 \div 1300 = 8.1$$

therefore

$$K' + 1 = 126{,}000{,}000 = 126 \times 10^6.$$

Since the 1 can be neglected

$$K' = 126 \times 10^6$$

and

$$\log K' = 8.1 = K_1.$$

Finally

$$T = \frac{10{,}500}{\log [1 + (126 \times 10^6/J)]} \qquad (6)$$

Again, as before, if we neglect the 1, equation (6) reduces to

$$T = \frac{10{,}500}{8.1 - \log J}$$

the familiar equation used in previous work.

Now when J is large, the 1 cannot be neglected, as will be shown. Let $J = 5 \times 10^6$, a rather large value, then

$$T = \frac{10{,}500}{\log (25.2 + 1)}$$

But $\log (26.2) = 1.41830$

and $\log (25.2) = 1.40140$

Consequently these two values will give temperatures as follows:

$T_P = 10{,}500 \div 1.41830 = 7400$ (Planck's formula)

and

$T_W = 10{,}500 \div 1.40140 = 7500$ (Wien's formula).

The use of Wien's equation will, therefore, introduce an error of about 1.3 per cent at 7400 deg.

Taking J as 4×10^4 there results

$$T = \frac{10{,}500}{\log [1 + (126 \times 10^6)/(4 \times 10^4)]}$$

In this case $\log (1 + 3150) = 3.49845$

and $\log (3150) = 3.49831$.

This difference will obviously not introduce any difference in the fourth place of the number representing the temperature.

$$T_P = \frac{10{,}500}{3.49845} = 3000$$

$$T_W = \frac{10{,}500}{3.49831} = 3000$$

Hence it is evident that for temperatures below 3000 deg. Wien's law is sufficiently accurate for all purposes.

If we consider a value of J even still larger than the above, say 100×10^6, the expression for the temperature becomes

$$T = \frac{10{,}500}{\log [1 + (126 \times 10^6)/(100 \times 10^6)]}$$

Here $\log (2.26) = 0.35411$

and $\log (1.26) = 0.10037$.

Whence

$$T_P = 10{,}500 \div 0.35411 = 30{,}000$$

and

$$T_W = 10{,}500 \div 0.10037 = 100{,}000.$$

Now there are stars which have temperatures as high as 30,000 deg. and evidently erroneous results would obtain if their temperatures were estimated by the aid of Wien's law. However, for such extreme temperatures even Planck's equation may be only a gross approximation.

Experimental Methods of Determining High Temperatures.

In the previous discussions on the application of radiation laws to the determination of temperatures the methods considered were not complicated by the introduction of the usual devices utilized for the estimation of very high temperatures.

Most of the pyrometers now in use will not indicate temperatures accurately beyond 2000 deg. unless the incident energy is diminished by means of some sort of an absorption device, such as absorption mirrors, absorption glasses, sectored disks, polarizing prisms, etc.

The ratio of the intensity of the light incident on such a diminishing device to the intensity of the light transmitted is called the *absorption factor*.

The intensity of the incident light is always obtained, therefore, by multiplying the transmitted intensity by the absorption factor. This use of the term *absorption factor*, in the case of transparent bodies, is somewhat different from that considered in the case of radiating bodies.

Let J' = the luminous intensity of the incident radiation and J the value as indicated by the pyrometer when an absorbing device is used, then

$$J' = JR,$$

where R is the *absorption factor*.

If, furthermore, this transmitted light passes through another diminishing device, whose absorption factor is R_1, then the intensity of the incident light must be expressed by the formula

$$J' = JRR_1$$

and in general

$$J' = JRR_1R_2 \ldots \ldots$$

or if the absorption factors are equal, then

$$J' = JR^n.$$

Suppose, for example, that the absorption factor of a piece of smoked glass is 20, if two such glasses are used to cut down the intensity of a luminous source, then the intensity of the incident light is 400 times as great as the intensity of the transmitted light.

Equation (4) may now be written

$$t = \frac{K_2}{K_1 - \log JR} - 273$$

or

$$t = \frac{K_2}{K_1 - \log J - \log R} - 273 \qquad (7)$$

Equation (7) may be applied directly to any pyrometer of the Wanner or Le Chatelier type for the determination of high temperatures when the absorption factor, R, is known. In the case of pyrometers which can only be calibrated empirically, such as the Holborn, Kurlbaum, the temperature T' of the radiating body in question, in terms of the temperature, T, indicated by the instrument and the absorption factor, R, may be determined from the following equation:

$$K = \frac{\log R}{K_2} = \frac{1}{T} - \frac{1}{T'} \qquad (8)$$

Total radiation pyrometers, such as the Féry and Thwing, are seldom used for the determination of very high temperatures, but when they are, the intensity is cut down by means of a sectored disk, *i. e.*, the effective aperture is simply made smaller.

As the total energy of the spectrum is utilized in this type of pyrometer any absorption device, such as smoked glass, absorption mirrors or polarizing prisms, which is selective in its absorption could not be used.

If J' represents the total energy falling upon the pyrometer and J the quantity indicated by the pyrometer when a sector disk is used, then

$$\frac{J'}{J} = R$$

where R is the absorption factor of the disk.

From the fourth power law

$$J' = KT'^4$$

and

$$J = KT^4$$

whence

$$J'/J = R = (T'/T)^4 \qquad (9)$$

By means of equation (9) the scale of any total radiation pyrometer may be extended when the absorption factor R of the disk is known.

The ratio of the intensity of the incident radiation to the

intensity of the transmitted radiation, in the case of the sectored disk, is obviously the same for all wave-lengths.

Absorption glasses, however, are usually selective in their absorption, *i. e.*, they absorb one part of the spectrum more than another, consequently the absorption factor of an absorption glass may be different for different wave-lengths.

Now, in any case where the light used by the pyrometer is strictly monochromatic, as is the case with the Wanner or Scimatco, an absorption glass will produce just as accurate results as any other diminishing device and it is considerably more convenient.

With the Le Chatelier, H-K and the Morse pyrometers, however, the case is quite different, for here the light is rendered monochromatic by means of colored glasses and these are never strictly monochromatic. The color transmitted by these so-called "red transmission glasses" will depend considerably upon the spectral distribution of the transmitting source, which in turn varies with the temperature.

If the wave-length of such a transmission glass is determined at two different temperatures of the radiating source, a decided difference will obtain. Consequently if the absorption factor R, of an absorption glass which is strongly selective, is determined for one temperature, the same factor cannot be used for any other temperature without introducing an error. Therefore Wien's law cannot be used directly for the extrapolation of the scale of such an instrument without making corrections for the lack of monochromatism and selective absorption. It is seen from equation (8) that the determination of T' involves R and also the wave-length λ, for the latter is included in K_2.

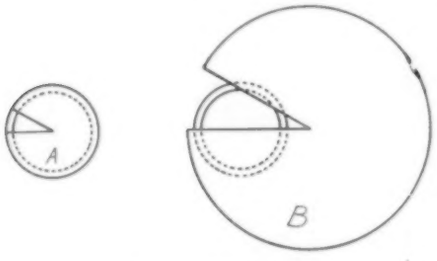


FIG. 1.—SECTOR DISKS.

Two types of sectored disks are shown in Fig. 1. A is a simple diaphragm provided with a sectored opening of definite angle, placed in front of the objective of the pyrometer. This is the sort of disk used on the Féry instrument.

B is similar to A, except that it is rotated by means of a small motor; moreover, the axis of rotation does not coincide with the axis of the instrument as in the first case. It is preferable to use the second type of disk in the case of the H-K pyrometer, since it will produce no distortion of the image which is essential in this type of instrument. Since the Féry instrument, when once focused on an object, functions independently, this fact is not so important, hence it is sufficiently accurate to use the more convenient stationary disk.

The absorption factor in either case is

$$R = \frac{360}{\phi}$$

where ϕ is the angle of the opening measured in degrees.

The absorption factor of an absorption glass, *i. e.*, of a so-called "smoked glass" is generally determined by means of a spectrophotometer. A spectrophotometer is simply a photometer which compares light intensities for a single color or wave-length. The absorption factor may also be determined by any pyrometer in which the intensity can be varied in a continuous and determinate manner as in the Wanner, Scimatco or Le Chatelier.

If a Wanner pyrometer is directed toward any constant source of illumination and a reading made without and with an absorption glass before the slit of the pyrometer, the absorption factor R may be calculated from the formula

$$R = \text{tang}^2\,\phi' \div \text{tang}^2\,\phi$$

ϕ is the angle through which the nicol is turned in order to obtain a photometric balance, when an absorption glass is used, and ϕ' is the corresponding angle obtained for a balance without an absorption glass.

If a Le Chatelier is used R then becomes

$$R = (s/s')^2$$

where s is the reading of the iris diaphragm with a glass before the objective and s' is the corresponding reading obtained without a glass.

The method of extending the scale of a pyrometer will now be illustrated by concrete examples of particular instruments.

High Temperature Scale of Le Chatelier Pyrometer

For a particular Le Chatelier instrument the pyrometer equation becomes

$$t = \frac{9700}{4.82 - 2\log(1/s)} - 273 \qquad (10)$$

See equation (3) of my article in the July issue. This equation can be modified by means of equation (7) to include the absorption factor. For low temperatures this instrument sometimes requires one or more absorption glasses before the comparison lamp, in which case log R is negative. Suppose that we use n absorption glasses, then equation (10) may be written

$$t = \frac{4.82 - 2\log(1/s) \pm n\log R}{9700} - 273 \qquad (11)$$

With this pyrometer directed toward a uniformly illuminated area of a frosted-globe incandescent lamp the reading s of the iris diaphragm was found to be 17.93 when one absorption glass was placed before the objective and 4.90 without any glass.

Hence

$$\log R = \log(17.93 \div 4.90)^2 = 1.13.$$

For $s = 20$ and $n = 0$.

$$t = \frac{9700}{4.82 - 2\log(1/20)} - 273 = 1033.$$

For one glass before the objective: $s = 25$, $n = 1$, $\log R = 1.13$,

$$t = \frac{9700}{4.82 + 2.80 - 1.13} - 273 = 1222.$$

For two glasses before the objective: $n = 2$,

$$t = \frac{9700}{4.82 + 2.80 - 2 \times 1.13} - 273 = 1520.$$

For one glass before the comparison lamp (n in this case is negative),

$$t = \frac{9700}{4.82 + 2.80 + 1.13} - 273 = 285.$$

In a similar manner a number of values of t were calculated and Table I constructed.

TABLE I.

Temperature in degrees Centigrade.

s	No glass	One glass (+)	One glass (—)
25	1000	1222	835
20	1033	1270	861
15	1080	1332	895
6	1247	1575	1020
4	1337	1712	1082
2	1513	1980	1210

High Temperature Scale of Wanner Pyrometer

For a particular absorption glass used on this instrument the reading of the analyzer was 17.47 deg., when the pyrometer was directed toward a luminous source of constant intensity. When the absorption glass was removed the analyzer had to be turned through an angle of 49.05 deg. in order to produce a photometric balance.

Therefore,

$$\log R = \log \frac{\tan^2 \phi'}{\tan^2 \phi} = \log \frac{\tan^2 49.05}{\tan^2 17.47}$$

$$= 1.123.$$

By means of equation (7) Table II was constructed.

TABLE II.

	Temperature in degrees Centigrade.			
	No	One	Two	Three
ϕ	glass	glass	glasses	glasses
10	910	1097	1358	1743
12	938	1127	1402	1810
..	....	....	....	....
..	....	....	....	....
..	....	....	....	....
..	....	....	....	....
78	1547	2053	2892	4752
80	1605	2148	3105	5314

In this case 45 deg. corresponds to a temperature of 1180 deg. C.

High Temperature Scale of H-K Pyrometer

This pyrometer must be calibrated empirically, since the intensity cannot be expressed by means of an equation. Its scale must, therefore, be extended by means of equation (8).

The wave-length of the red glass used before the eye-piece was determined by means of a Koenig spectrophotometer and found to be 0.650μ. For this wave-length the absorption factor was determined by means of the same instrument. Log $R = 1.597$. Hence K of equation (8) becomes

$$K = (\log R) \div K_2 = (1.597 \times 0.650) \div (14{,}500 \times 0.4343) = 0.000166.$$

By means of the factor K the values of t' in Table III were determined. For example, suppose we have a calibration for the pyrometer, then we may assume various values of temperature from this calibration and calculate the corresponding temperature that will give the same intensity when an absorption glass is used.

From equation (8)

$$1/T' = (1/T) - 0.000166.$$

For 800 deg. C. or 1073 deg. absolute,

$$1/T' = (1/1073) - 0.000166.$$

Whence $T' = 1305$.

Therefore $t' = T' - 273 = 1032$.

TABLE III.

t deg. C.	T deg. abs.	T' deg. abs.	t' deg. C.
800	1073	1305	1032
1000	1273	1615	1342
1200	1473	1948	1675
1400	1673	2315	2042
1600	1873	2715	2442

A curve may now be plotted with the new values of temperature, column 4, and the current values corresponding to the old temperatures in column 1 (Table III).

This pyrometer has been greatly improved by Mendenhall for precision work. In its modified form the red transmission glass is replaced by a spectroscope eye-piece which renders the light used strictly monochromatic. The intensity is cut down by a rotating sector disk actuated by a small motor attached to the frame of the instrument.

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The Electric Smelting of Zinc Ore[1]

BY WALTER RENTON INGALLS.

The electric smelting of zinc ore during recent years has engaged much attention. It is not a new idea. The Cowles brothers tried it in 1885, which was the first attempt (so far as I am aware), and since then numerous experimenters have undertaken it. It is not my intention in this paper to trace their work, or go further into the history of the subject. It is sufficient for my present purpose to mention that C. P. G. De Laval[2] was the first to erect an electric zinc-smelting furnace on a commercial scale, this having been done in Scandinavia in 1901. Works were later erected at Sarpsborg, in Norway, and at Trollhättan, in Sweden. I do not know precisely the dates of their erection, but they were in operation as early as 1905. It was attempted at first to smelt ore, but for a long time this was not commercially successful, the product being chiefly zinc powder, which had to be subjected to resmelting, giving an impure spelter, which had to be refined. The works were continued in operation in a more or less irregular way and spelter from them came on the market, but this was the product of smelting dross, scrap and other secondary products rather than ore. Some progress in ore smelting was made, however, and during 1911 a considerable quantity of ore was smelted by a method to which I shall refer subsequently.

There is much misconception respecting both the ordinary art of zinc smelting, and the possibilities of electric smelting. In the first place, the ordinary art of zinc smelting is not extraordinarily backward, in spite of the small retorts to which we are limited; and although we are attracted by the possibility of conducting the distillation in a relatively large apparatus, electrically heated, I am not yet by any means prepared to say that electric smelting will offer any advantage over existing methods, even if the difficulties that are now in view can be overcome. In the second place, electric smelting is not to be expected to accomplish the establishment of any radically new metallurgical principle. If there be any dreams of an electric furnace analogous to the blast furnace, in which run-of-mine ore may be charged roughly and from which spelter, lead and perhaps also copper matte, may be obtained concurrently, I fear that there will be no realization of them.

Carbon Reduction and Iron Decomposition.

Whatever be done in electric smelting is generally expected to be based on the reduction of zinc oxide (or its compounds) by carbon (and carbon monoxide); but the decomposition of zinc sulphide by iron has also been contemplated. The former is the basis of ordinary practice. The latter has been exploited in the electric smelting of Coté-Pierron[3] and of Imbert-Thomson-Fitzgerald.[4] While the decomposition of zinc sulphide by iron has not found application in ordinary furnace work, it has been repeatedly suggested. Indeed, the Imbert process was originally such a suggestion, the use of an electric furnace for its execution being an afterthought.

The Coté & Pierron process was tried experimentally at Lyon, France, in 1906, and in 1907 at Arudy, in the Basses-Pyrenees, and was ephemerally the subject of many articles in the technical press, but lately it has not been mentioned, and I am inclined to think that the results were not satisfactory.

The Imbert-Thomson-Fitzgerald furnace has been in trial at Hohenlohehütte, Upper Silesia, during the last two years. No authoritative reports of the results recently obtained there have been made public. The imperfect news that seeps through professional and commercial channels is conflicting.

I shall not in this paper discuss either the Coté-Pierron or the Imbert-Thomson-Fitzgerald proposals, but shall confine myself to a consideration of the reduction of zinc oxide by carbon. In this discussion I shall draw upon general metallurgical experience, upon the information obtained in experimental electric zinc-smelting at McGill University, and upon the authentic reports of work that has been done elsewhere.

The reduction of zinc oxide is an endothermic process, i. e., heat must be supplied to it. This may be done either by the external heating of the retort containing the charge by the combustion of coal, or by internal heating by means of the electric current. The theoretical advantage of the latter rests in the possibility of developing the heat in the midst of the charge, i.e., in the heart of the place where the work is to be

[1] A paper presented at the meeting of the Canadian Mining Institute at Toronto on March 8, 1912.

[2] See this journal, Vol I, 425 (1903), II, 423 (1904), III, 54, 209 (1905), V, 241 (1907).—ED.

[3] See this journal, Vol VII, 468 (1909), VIII, 99 (1910).—ED.

[4] See this journal, Vol. VIII, 289, 317 (1910), IX, 259 (1911).—ED.

done, a condition that does not exist in ordinary external heating.

The electric furnace may be of two types, namely, the arc furnace and the resistance furnace. To draw a familiar analogy, the arc furnace is exemplified by the ordinary arc lamp used for illumination; the resistance furnace by the ordinary incandescent lamp. Another type of furnace, namely, the induction furnace, will not here be taken into consideration.[5]

Types of Furnaces.

The original De Laval furnace was a simple arc furnace, in which the heat of an electric arc, formed between electrodes projecting horizontally into the furnace, acted upon the charge of ore lying under it. Among the various types of resistance furnace we may have that in which the current passes through a resistor of carbon (as in the Imbert-Thomson-Fitzgerald), or that in which the electrodes dip into the charge to be smelted, or into a bath of slag which serves as the resistor. It will be unnecessary for the present consideration to go any further into the subject of the types and design of furnaces. It is the fundamental metallurgical principles upon which it is still most important to center attention. In the present status of the art of electric zinc-ore smelting, the functions of the electrician are subordinate to those of the metallurgist.

In the earliest attempts to smelt zinc ore electrically, the conception was of an apparatus similar to an ordinary retort, but electrically heated. In some of our early experiments at McGill University, we used furnaces of analogous character. The rate of reduction in these furnaces was about on a parity with the rate in the ordinary retort, temperatures being equal, which was, of course, what was to be expected.

These experiments, made primarily for another purpose, confirmed the idea that if electric smelting were to offer any advantage, it must be conducted continuously. Here is introduced a departure from the conditions of ordinary smelting, which is an intermittent operation, conducted in two stages, the first comprising the expulsion of aqueous vapor, and the decomposition of hydro-carbons and carbonates; and the second being that of the reduction of the metallic oxides.

If zinc smelting is to be done continuously, it seems that the operation must be confined to the second stage, which implies that the things that ordinarily happen in the first stage must be caused to take place by a previous pre-heating. With such a preheating the actions in the furnace become essentially the same, save for certain important differences.

Differences from Ordinary Practice.

One of these differences pertains to the carbonaceous reduction material that is mixed with the ore. It has been shown by Boudouard that the permissible limit of carbon dioxide in the gas of the zinc retort is very low; varying with the temperature, but still very low at the very high temperature of 1500 deg. C., a temperature that is not attained in practice. When the permissible limit is exceeded, carbon dioxide acts oxidizingly upon the zinc, and coating the condensing globules of the latter prevents coalescence and forms powder.

The gas issuing from the zinc retort of practice is within the permissible limit of carbon dioxide, or nearly within it, and the zinc is condensed chiefly as molten metal. Conversely, all attempts to smelt zinc ore in the blast furnace have resulted only in the production of powder because of inability to control the percentage of carbon dioxide. Small quantities of molten zinc have been occasionally found in the blast furnace, undoubtedly due to the accidental occurrence of favorable conditions.

The uniformly favorable conditions of the ordinary zinc retort are due to its small size, its confined character, and the presence of an immense excess of carbon in the charge. Carbon dioxide, resulting from the reduction of zinc oxide, and from other reactions, is immediately exposed to the great surplus of finely divided, intimately admixed, incandescent carbon, and is reduced by it to monoxide.

Continuous charging implies discharging of the residue without interrupting the smelting operation. This in turn implies the discharging of the residue as a slag that will run. Both conditions imply the absence of any great excess of carbon, which cannot be slagged and cannot be permitted to accumulate in the furnace. In this important particular, the internally heated, continuously operated retort or furnace differs essentially from the externally heated, intermittently operated retort.

Carbon Dioxide.

In our experimental work at McGill University, we have commonly found high percentages of carbon dioxide in the gases from the furnace, much higher than exist in the gases from the ordinary retort. Besides this difference we have observed the presence of considerable quantities of carbon and ore-dust in the condensers, indicating another variation from the conditions of the ordinary retort. In the continuous feeding of the charge into the electric furnace, there is naturally stirred up some dust which may be, and probably is, carried out into the condenser by the relatively strong outrush of gas and vapor; the latter may indeed entrain some dust from the surface of the charge in the furnace. There may be some other physical disturbances.

The effect of the carbon dioxide in the gas has been recognized by Coté & Pierron, by Johnson,[6] and by other investigators who have caused the gas and vapor from the retort to pass through a column of incandescent carbon, the purpose of which is to act as a secondary reducer. This is believed to be, at least, ameliorative.

Except for the metallurgical differences that have been previously pointed out herein, the reactions and condition in the internally heated retort are essentially the same, providing that the charge introduced into the former be preheated to a degree corrsponding to the preheating that takes place in the first stage of the intermittent process. The gradually introduced charge, with its interstitial air, should not involve any new conditions, except those previously noted, inasmuch as the gas inside of the furnaces displaces the interstitial air before the charge enters into the smelting chamber. The reduction of lead and iron, developing carbon dioxide, and the multiplicity of complex side-reactions, take place in one kind of retort as well as in the other.

There is no positive evidence that the electric current passing through a furnace plays any metallurgical part other than thermic. Harbord reports no observed difference between the actions of the arc furnaces at Sarpsborg and the resistance furnaces at Trollhättan, except in the efficiency of the energy supplied.

At McGill University we observed arcking in the furnace when a resistor of fragments of carbon was used; and this arcking was followed by disturbance in the furnace that we have been unable to explain; but there was no such occurrence in a furnace with a continuous resistor or carbon, in which furnace the reduction of zinc oxide appeared to take place regularly and normally. This particular furnace was provided with a peep hole (closed by mica), through which the process of distillation could be observed. The resistor was vertical in position, the charge surrounding it. The distillation proceeded most actively around the resistor, the heating element, the charge coning down toward this as the zinc was expelled, and the interior of the furnace maintaining an even incandescence.

Speed of Reduction.

Our attmpts at McGill University to reduce zinc oxide in an intermittently charged, electrically heated retort, resulted in

[5] An induction furnace has been proposed by G. Gin, but it has not been tried, so far as I am aware. [For description of this furnace see this journal, Vol VI, 75 (1908).—Ed.]

[6] See this journal, Vol. VIII, 688 (1910), IX, 264 (1911), X, 281 (1912).—Ed.

so slow a reduction that the prospect of advantage seemed to be insignificant. I refer to a retort in which the charge was introduced in the usual manner at the beginning of the operation, the residue to be discharged as an unfused ash. There was, of course, a possibility of converting this from an intermittent to a continuous operation, but, as I have already stated, the slowness of the reduction under these conditions dispelled any such idea.

When we changed to the scheme of starting the furnace with a bath of slag, continuously introducing the charge upon this bath, we found immediately that the reduction became relatively rapid. I am unable to offer an adequate explanation of the improvement in this respect. I can but state that a retort of small proportions smelts a surprisingly large quantity of charge by comparison with the retort of ordinary practice at an equivalent temperature.

With a furnace with a plumbago crucible 12 in. in diameter, we have been able to smelt 10 lb. per hour. With a furnace 18 x 18 in. in horizontal section we smelted 737 lb. in 23 hours. We have learned consequently that the claims of extraordinary capacity of electric zinc smelting furnaces of comparatively small size are well founded.

Bearing in mind the main difference between the conditions of ordinary and electric zinc smelting, we may pass on to a consideration of similarities. There is no reason to believe that the reduction of the metallic oxides, the reactions between metals, oxides, sulphides and sulphates, the scorification of oxides, and, in short, the multiplicity of complex reactions that we believe to take place in the externally heated retort, will be any different in the internally heated, the nature of the charge and the temperature remaining the same.

The Matter of Temperature

With respect to the matter of temperature, that of the electric furnace may be anything that it is pleased to make it, within the limitations of the material whereof the furnace may be constructed. In the ordinary practice of zinc smelting, the tendency in Europe during recent years has been to drive the furnace harder and harder.

A furnace temperature of 1400 deg. C. was once thought to be high, but now 1500 deg. C. and upward is reported. Such high temperatures imply, of course, the necessity for much more substantial furnace construction and a high class of refractory material. Of course, the average life of the retorts is reduced.

The actual smelting temperature, i.e., the temperature inside of the retorts, is about 100 to 200 deg. less than the furnace temperature. In this particular the electric furnace has the advantage that the full temperature of the furnace is ralized in just the place where the smelting is to be done.

The amount of zinc remaining in the residuum, whether the latter be an ash (loose or clinkered) or a slag, depends primarily upon the temperature of the smelting and the fluxing of the ore. Conditions favorable to low zinc retention are high temperature and highly calcareous slag. It is possible to discharge from the electric furnace a slag containing less than 1% of zinc. We have made such slags at McGill University and other experimenters also have done so.

The ability to do that does not, however, imply either commercial feasibility or advisability. It may be impossible to maintain a furnace in continuous operation under the conditions necessitated by the production of such a slag. This matter has not yet been worked out. The slag made at Trollhättan, during Mr. Harbord's test, was high in zinc.

We do not as yet know much respecting the shape that the smelting chamber of an electric furnace will naturally assume, the formation of refractory accretions on the walls, the deposition of sows of metallic iron (that doubtless will be produced in the same way as in the ordinary retort), and the difficulties that are likely to occur in continuous operation from these things.

Smelting at Trollhättan.

The operations at Trollhättan and at Sarpsborg were the subject of a report by F. W. Harbord, in 1911, which is the only recent source of technical information respecting them. Mr. Harbord's report was made privately for commercial purposes and did not go fully into the metallurgy of his subject. Mr. Harbord conducted a test at Trollhättan, during which 537 metric tons of ore were smelted, the test extending through 27½ days. This is the longest continuous run on a large scale that has been reported.

The ore was essentially a zinc lead ore, rather than a zinc ore, and was high in silver, containing 36% zinc, 24% lead and 30 oz. silver per ton. In making deductions from the experience in smelting this ore, we must hesitate in applying them too freely to the cases of what are more distinctly zinc ores. Mr. Harbord extracted 64% of zinc, 74% of lead and 46% of the silver as metals, which percentages would be slightly increased by further treatment of certain incidental products, but even so the metal recovery would still be relatively low. A summary of Mr. Harboard's report is to be found in the *Engineering and Mining Journal* of February 10, 1912.[1]

Briefly, the furnaces were of the resistance type, having one large vertical electrode passing through the roof, the other electrode being a carbon block bedded in the bottom of the furnace. Each furnace smelted about 2800 kg. of ore per 24 hours. The product was partly metal and partly blue powder and oxide, containing about 54% of zinc and 20% of lead.

This powder is mixed with fresh ore and recharged, when a larger percentage of the metal volatilized is recovered as metal. The first operation of smelting ore alone is to be regarded mainly as a concentrating process for the production of a rich oxide.

The regular charge was 100 kg. of roasted ore, 200 kg. of powder, 25 kg. of coke dust and 5 kg. of lime. Thus for every ton of ore smelted, two tons of powder had to be resmelted; indeed rather more than that, there having been an accumulation of powder during the run.

Recovery of Lead.

In the ordinary retort the lead content of the ore is reduced and is partly volatilized, condensing with the zinc, which it contamintes; and partly remains as metal in the residuum, from which it is recovered by smelting the latter, or by jigging and smelting the concentrate. The percentage of lead volatilized increases with the temperature. In the electric furnace lead behaves in the same way, but in fusing the residuum into a slag the lead is given an opportunity to separate and thus may be tapped separately, which in so far is an advantage. On the other hand, the desirability or necessity of smelting at a high temperature leads to the collection of a leady spelter, which must undergo a subsequent refining, and the experience at Trollhättan has indicated difficulties in the furnace construction because of the lead.

I am not greatly attracted by the possibility of smelting for lead and zinc in a single operation, much less by aiming also to make a matte (although that may be inevitable), which adds to the difficulties, nor do I see any good reason to head in that direction. In most kinds of mixed sulphide ores the galena is easily separated to a fair degree by gravity concentration, and no loss need be experienced in such a separation, inasmuch as the galena and the blende-pyrites may be easily collected in filter bins. It should be the policy first to subject the ore to this cheap process, very likely cutting out a portion of the gangue sufficiently clean to be discarded, and pass the galena on to ordinary lead smelting, also a relatively cheap process, rather than to deliver all of the ore to the zinc-smelting process, which must inevitably be relatively dear, whether it be the ordinary one or the electrothermic.

There will be, of course, from such a treatment a percentage

[1] Also in this journal, Vol. IX, 673 (1911).—ED.

of lead in the blende-pyrites, and being there it will be desirable to extract it, but it is troublesome both in the roasting and in the smelting, and I conceive that the electric smelter, like the ordinary smelter, will rather have a zinc ore without lead than with it, unless it can be obtained on terms that compensate for the disadvantages. It may be accepted, however, that the electric furnace will probably obtain a higher extraction of the lead and at less cost than is possible in the ordinary method of smelting, and there are some ores wherein the galena and blende are so intimately mixed that mechanical separation is impossible, which may be treated more advantageously in the electric furnace than in any other way. I do not by any means regard this as a step in metallurgy that has yet been accomplished, but it is a possibility. The work at Trollhättan has been in this direction.

Condensation.

It has been assumed by some investigators that the difficulty of condensing the vapor resulting from the reduction of zinc oxide, lies chiefly in the condensing apparatus. It may be taken for granted that the condenser connected with an electric zinc-smelting furnace, required to deal with a relatively large quantity of vapor and operating under some conditions different from those existing in connection with the ordinary furnace, must be of dimensions and design conforming to the new conditions.[1]

It has been found possible to condense the zinc satisfactorily as spelter from the electric furnace treating zinc powder and galvanizers' dross. This had been done at Trollhättan, where, in the refining of crude zinc, but 9.8 per cent of powder has been collected. In refining 111,794 kg. of leady spelter, Mr. Harbord obtained only 11,209 kg. of powder. Charles A. Weeks reported, in *Metallurgical and Chemical Engineering* of July, 1911, that in the refining of zinc dross in an electric arc furnace, during a run of 36 hours, no powder was made. In a private communication, Mr. Weeks informed me that this furnace was built and operated in the works of the General Electric Company at Schenectady, N. Y.

Accepting the foregoing evidence, the reason for the production of blue powder is to be sought not in the condenser proper, but in the character of the gas and vapor delivered to the condenser; in other words, to the conditions that exist and the reactions that take place inside of the furnace itself. It has been found that when metallic zinc is simply distilled, there is no great difficulty in condensing the vapor in a liquid form, but when the vapor originates from the reduction of zinc oxide, there is difficulty.

Nearly all experimenters in ore smelting by the reduction of zinc oxide by carbon have been baffled in the attempt to condense a commercial proportion of the zinc as molten metal. We have encountered this difficulty in our experimental work at McGill University, and have not yet been able to master it. The reason why zinc vapor from the electric furnace should condense in this way has not yet been satisfactorily explained. It may pertain to the differences between the conditions of ordinary smelting and electric smelting that I have already mentioned; it may pertain also to other conditions. I shall not advance hypotheses respecting this; I shall limit myself to the mere statement of the fact.

Blue Powder.

At the present time the matter of reducing and distilling the zinc and condensing the latter chiefly as spelter and blue powder, is the crucial point of electric zinc-ore smelting. Until that has been accomplished, I can see no future for the process, except under special conditions of extraordinarily cheap power, peculiar kinds of ores, etc.

[1]The condenser of the electric furnace receives no radiant heat from the front of the furnace nor from the glowing charge in the retort. On the other hand, the condensation of a large amount of zinc vapor liberates a relatively large amount of latent heat. The balance between these conditions has by no means been worked out. We find some experimenters having artificially to cool the condenser and others to heat it.

On the basis of ore alone, the consumption of power at Trollhättan was 2078 kw.-hr. per 100 kg. of ore, but for every ton of ore smelted about two tons of powder had to be resmelted. The smelting of 1000 kg. of ordinary zinc ore is reckoned to require from 900 to 1000 calories of heat, which is equivalent to 1047 to 1163 kw-hr. of electric energy, these figures representing the cases of ordinary ores containing 25 to 50 per cent zinc. In one run at McGill University, lasting 24 hours, with a resistance furnace, 18 x 18 in., Dr. Stansfield determined the power consumption per 2000 lb. of ore smelted as 1610 kw.-hr. (about 1771 per 1000 kg.).

It has been claimed that it is possible to smelt 2000 lb. of zinc ore with 1000 kw.-hr. While it is doubtful whether such a result has yet been actually realized by anyone, the claim is plausible with respect to certain kinds of ore, although even then it is necessary to assume a high furnace efficiency. It would seem that 1200 kw.-hr. per 2000 lb. of ore would be sufficiently brilliant realization. Even such a one would put electric smelting upon a sound basis in so far as power consumption is concerned.

Cost of Smelting.

The art of electric zinc smelting is still too much in its infancy to permit of much more than speculation respecting the probable cost. The only data of operation on a large scale continued for a considerable period of time that are available are those of Trollhättan, where electric power, labor, etc., are obtained very cheaply. If the Trollhättan figures be applied to the conditions of power at $20 per horsepower per annum, labor at $3 per eight hours, and other elements in proportion, electric zinc smelting on the basis of the metallurgic results obtained there would be out of the question. If it becomes possible to smelt with 1200 kw.-hr. per ton of ore, to reduce materially the labor and electrode costs, and to extract 90 per cent of the metals, there will be some chance.

As to the probable cost of plant we are still greatly in the dark. It is hardly to be expected that the electric plant will be of any less cost than the coal-fired. Comparing operating costs, there will be the same expense for administration, general charges, yard labor and roasting in each case. Heating coal will stand against electric current, say 1 to 1.5 tons against 1200 kw.-hr. The relative consumption of reduction coal is uncertain. The retorts of ordinary practice will be dispensed with, but instead of them the electric smelter will have an electrode expense. The relative cost of repairs and renewals and miscellaneous supplies is uncertain; also the matter of labor.

At first thought it seems that the labor on a furnace smelting 6200 lb. of ore per day ought to be less than in connection with 80 Rhenish retorts or 150 Belgians, but the experience at Trollhättan does not yet confirm that idea, though, no doubt, improvement will be realized in this particular.

Limitations.

In the report of the Commission to Investigate the Zinc Resources of British Columbia, published in 1906, I made the following remarks:

> The successful smelting of zinc ore in the electric furnace is a matter which depends chiefly, indeed I may say solely, upon the cost of the process. De Laval has shown that it can be done. The doing of it profitably is quite another question. This is dependent chiefly upon the cost of power, the electrical efficiency of the furnace, and finally upon the purely metallurgical difficulties that pertain to the reduction of zinc oxide and the condensation of the metallic vapor.
>
> From the present development of electrothermic smelting of zinc ore, and the calculations respecting it, the following conclusions may be drawn:
>
> 1. Electric smelting will never displace ordinary smelting, if it be necessary to generate the power from coal.
> 2. Electric smelting may be, in the future, economically conducted at places where very cheap hydro-electric power is available.
> 3. Aside from the question of power, up to the present time certain peculiar and serious metallurgical difficulties in electric smelting have not been satisfactorily overcome.

During the six years that have elapsed since those words were written, much has been learned respcting their subject, but as generalizations they remain as true to-day as they were then.

The successful achievement of smelting zinc ore in an electric furnace will probably be the result of learning how to run the furnace; i.e., how to control the process, rather than of any distinct invention. I conceive that the electric zinc-ore smelting furnace will be an apparatus of general character, such as a blast-smelting furnace for lead ore, etc. This is to say: Lead ore may be successfully smelted in many forms of furnaces of the same generic character, but with variations in their lines that are of important economic effect. Once it has been learned how to run an electric furnace for zinc smelting, I believe it will be found that the process can be conducted in many forms of furnaces conforming to the general type, wherein the principle will be the same although the economic results may differ widely.

Electrical Resistances of Powdered Conductors—Variation with Temperature

BY A. A. SOMERVILLE.

Contact resistance is a source of trouble in making electrical measurements. If two materials are fastened together the surfaces must be clean and sufficient pressure applied to render the contact resistance at the joint a negligible quantity.

A study of this phenomenon was thought desirable and carbon and silicon were chosen as favorable materials to investigate.

In order to make the contact resistance as prominent as possible the materials were finely powdered so as to make a large number of contacts.

The powder was of such a fineness that it would pass through a screen having forty meshes per inch. A porcelain tube was

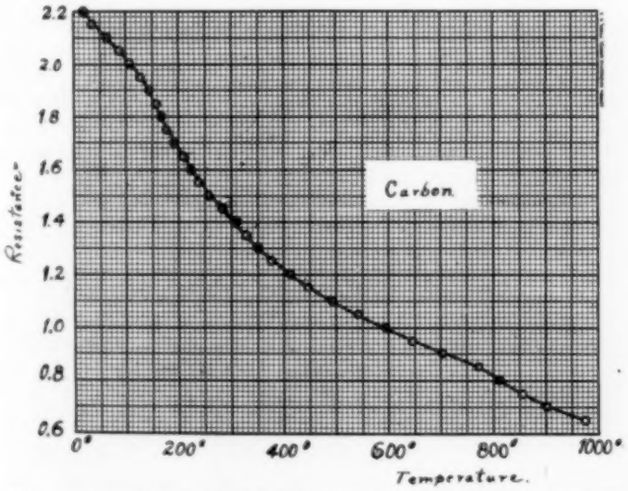


FIG. 1.—CHANGE OF ELECTRIC RESISTANCE OF POWDERED CARBON WITH TEMPERATURE.

filled with this powdered material and metal rods inserted into either end of the tube. The rods served as electrical leads or terminals and considerable pressure could be put upon the powdered carbon or silicon by transmitting the force through the metal rods.

It was found that there was practically no change in resistance after a pressure of about 850 pounds per square inch was reached. The resistance had decreased quite rapidly as pressure had been incresed up to that value. This part of the work has been described in the *Electrical World* of May, 1912.

Since then the work of measuring the variation of electrical resistance with temperature has been done.

The powdered material was placed in the porcelain tube as usual with the metal rods inserted in either end of the tube and the whole inserted into a tubular electrical resistance furnace. A pressure of about 3 lb. was put upon the rods and this pressure was held constant while the temperature of the furnace was varied from 20 deg. to 1000 or 1100 deg. C. The temperature was measured with a platinum resistance thermometer which lay alongside the specimen of carbon or silicon and the varying temperature indicated by an automatic temperature recorder to which the thermometer was connected.

Resistance of the specimen was measured on a dial bridge. Resistance of the lead wires was negligible. The length of the powdered specimen in the tube was 5 cm; cross section was 1 sq. cm.

At room temperature the resistance of the powdered carbon was 2.2 ohms; resistance of the silicon, 600,000 ohms. This is

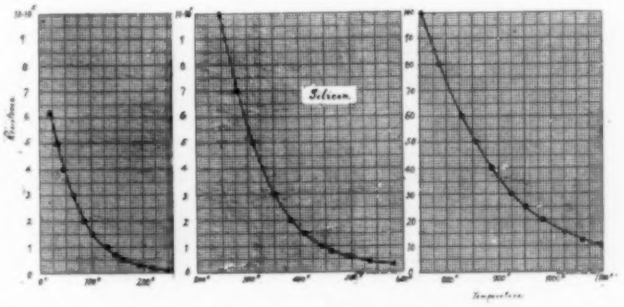


FIG. 2.—CHANGE OF ELECTRIC RESISTANCE OF POWDERED SILICON WITH TEMPERATURE.

approximately fifty times as much as the resistance of a solid piece of carbon of similar dimensions and 25,000 times as great as that of a similar piece of solid silicon.

The resistance of the carbon decreases until it is about .6 of an ohm at 1000 deg. C. The slope of the curve is exactly the same as that for a solid piece of carbon but the rate of change is more than twice as great when pulverized than when solid.

In the case of the silicon the decrease in resistance is much more rapid. In order to show the variation to advantage it is necessary to plot the different parts of the curve to different scales. The variation in resistance of a solid piece of silicon is unique being first negative then changing to a positive value and later to negative again, but the whole variation is always less than 50 per cent of the value at any time.

Ithaca, N. Y.

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Open-Hearth Refining.—Heating with coal dust has been practised in open-hearth refining, but according to the claims of GASTON BARBANSON and MAX LEPERSONNE, of Brussels, Belgium, the methods in use have not profited by all the advantages of this mode of heating. The method of using highly heated air for the injection and combustion of the coal dust has resulted in partly gasifying the dust before ignition of the flame, thus giving gas-heating with so large a flame that it could not be concentrated on selected portions of the bath of metal. When working with cold air combustion chambers are necessary, and the heat being generated in a comparatively large space cannot be concentrated as desired. For these reasons the electric furnace has proved the most practical means of applying heat in a concentrated form for steel refining.

In order to utilize coal dust for the generation of heat, which can be applied as desired and in a manner approximating the same results to be obtained in the electric furnace, Barbanson and Lepersonne have invented means of injecting coal dust in a manner which overcomes the objections first mentioned. The nozzle through which the air and coal dust are injected is situated as short a distance from the bath as is feasible, the distance being a function of the mean length of the flame obtained, which may advantageously be about 40 in. The air used is not

highly heated, but only slightly preheated, in order to avoid the gasifying of the fuel as noted above. To obtain the highest possible temperature the two substances are injected in correct proportions for complete combustion, although for special purposes either air or fuel may be in excess. The flame impinges upon the bath so that the heat of combustion will act directly upon the latter. By this arrangement the inventors believe that more economical results will ensue than can be obtained from electric refining, and that the metallurgical results will be equally as good. (1,030,152, June 18, 1912.)

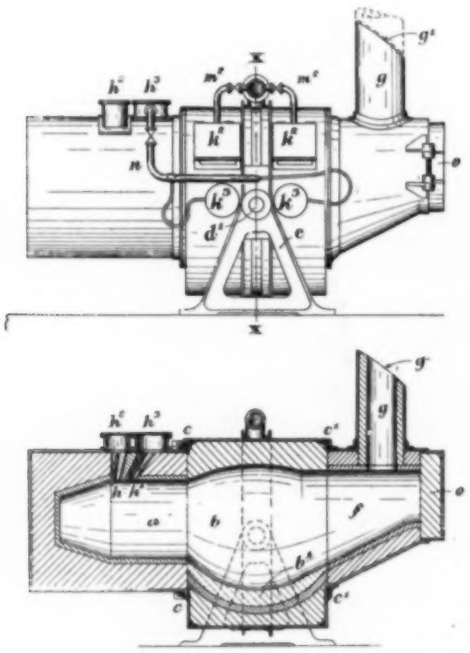


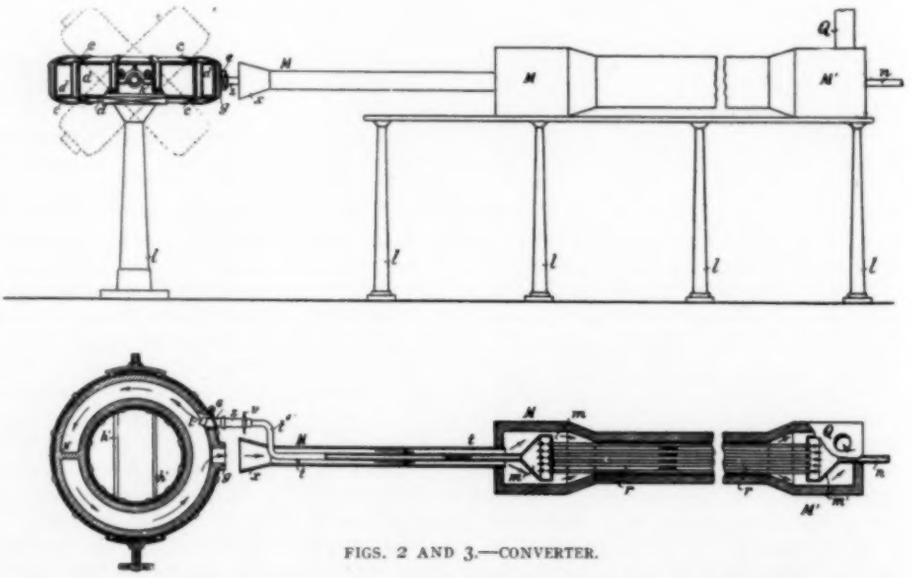
FIG. 1.—COMBINATION OF CONVERTER AND REVERBERATORY.

Manufacture of Steel.—Special apparatus designed to insure the production of castings free from blow holes, has been patented by T. D. Mackie and G. F. Forwood, of Limpsfield, England. In the production of steel by the Bessemer process there is tendency for some of the iron to become oxidized, with the result that blow holes occur in finished castings. According to the present invention, the oxidized metal and slag from the converter are transferred to the hearth of a reverberatory furnace, and, after the removal of the slag, the metal is subjected to the action of a reducing gas, with the result that it becomes deoxidized.

In Fig. 1 is shown the apparatus designed by the inventors, consisting of a combination converter *a* and reverberatory *b*. The two parts are suitably joined at the line *cc*, and the whole apparatus is mounted on trunnions d^1, about which it may be rotated into either a vertical position for converting or a horizontal position for the second process of deoxidizing. Tuyeres for air and producer gas are located at *h* and h^1 respectively. The furnace is first heated while in the horizontal position, and then swung into a vertical position to receive a charge of metal. A blast of hot air is then introduced through the tuyere *h* or both *h* and h^1, and the blowing carried on until the percentage of silicon and manganese required in the finished product is approximately reached. The air blast is then shut off, and the converter swung into a horizontal position, bringing the metal onto the reverberatory hearth *b*. Air and producer gas are again admitted and adjusted to produce a long flame over the surface of the liquid. The charge on the hearth is now treated with oxide of iron as in an open hearth furnace, and after the slag has been thickened by the addition of suitable material, the furnace is tilted slightly to facilitate the removal of the slag. The furnace is again returned to the horizontal position, and the metal is exposed to the action of a hydrogen-carrying gas, containing 50 per cent hydrogen, nitroduced through the tuyere h^3, while an air supply is admitted through the higher tuyere h^2. The gaseous mixture ignites at the top of the furnace and maintains, by radiation, the temperature of the bath. Thus, although an oxidizing condition prevails at the top of the furnace, a stream of deoxidizing gas impinges on the surface of the metal and removes oxygen present in the metal. After this process has been completed, the usual additions of carbon, manganese or silicon can be made, and the charge is then ready for pouring. (1,024,909, Apr. 30, 1912.)

Converter.—An improved type of converter for turning pig iron into steel has been patented in the United States by Srol B. Frumkin, of Minsk, Russia. As illustrated in plan in Fig. 2 and in elevation in Fig. 3 it consists of an annular furnace hearth, which is the converter, connected with a heat regenerating apparatus in which the hot gases issuing from the converter yield their heat to the incoming air blast. The converter is placed on trunnions so that it can be tilted or revolved through an angle of 180 deg. if desired, for the purpose of receiving or discharging metal. Suitable openings are made in the body of the converter to admit fresh air and discharge the products of combustion, the former coming from the regenerator and the latter passing through it. The passage of air through the converter is shown by the arrows.

The inventor claims the following advantages for his converter: 1. Loss of iron is avoided by directing the blast toward the surface of the metal instead of through it as in the Bessemer process. 2. The gases developed from the metal bath are burned, converting CO into CO_2, thereby increasing the temperature. 3. The gases act on the surface of the bath for a much longer time than in the Bessemer process. 4. Owing to absence of CO and increased percentage of CO_2 in the fire gases coming in contact with the molten metal, sulphur and other elements having passed into the slag are not able to re-



FIGS. 2 AND 3.—CONVERTER.

duce and return to the molten metal, as is possible in the Bessemer process. 5. Owing to the intense heating of the chamber and particularly the roof and upper walls, it is possible after the termination of the process to turn the apparatus through an angle of 180 deg. and allow the metal to rest on the glowing lining for a time, whereby absorbed gases can be driven out and the metal rendered more dense. At the same time the extraction of any FeO is completed by reaction with the contained carbon. 6. Cost of construction and operation is cheaper than with the Bessemer process.

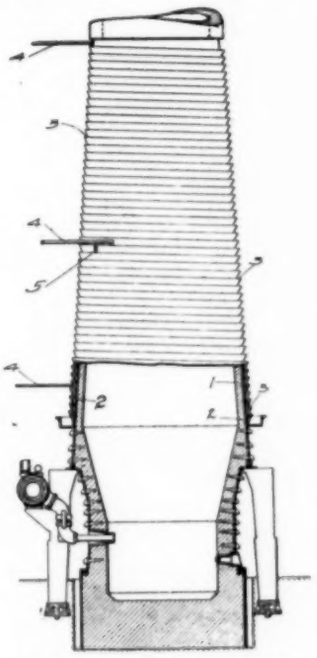


FIG. 4.—FURNACE COOLING DEVICE.

In addition to using the device as a converter the inventor also claims that it is an efficient device for roasting and reducing iron ores to metal. (1,025,260, May 7, 1912.)

Blast Furnace Construction.—A novel method of cooling thin-walled blast furnaces is proposed in specifications of a patent granted to Mr. FRANK C. ROBERTS, of Wynnewood, Pa. As shown in Fig. 4, the idea consists in surrounding the outer shell of the furnace with a helicoidal trough formed by a strip of sheet metal fastened to the furnace shell. The water used for cooling is delivered into the trough at various places from pipes 4, and flows down the spiral trough to collecting launders at the bottom. The bottom of the spiral trough may be more or less open along its bottom, so that the water need not follow a spiral path only, but a portion of it may pass vertically from trough to trough, thus covering the surface of the furnace. (1,030,792, June 25, 1912.)

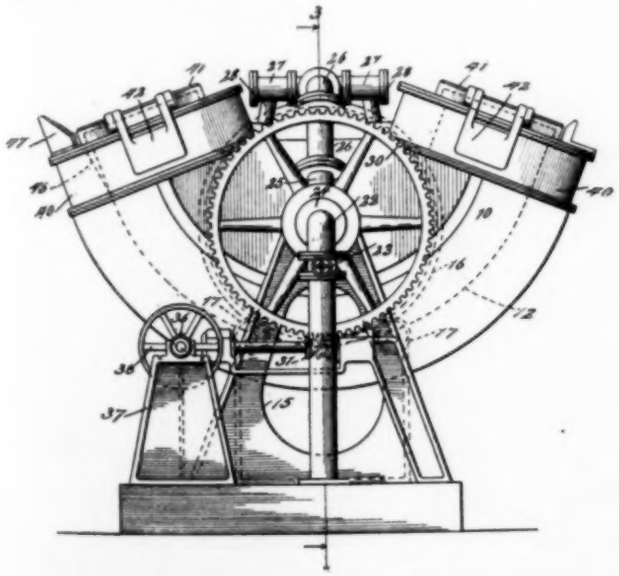


FIG. 5.—FURNACE FOR BALLING SCRAP METAL.

Balling Scrap Iron.—In Fig. 5 is shown a type of furnace designed and patented by MORRIS SEELERS and HAL M. NICHOLLS, of Chicago, Ill., to be used in welding scrap iron in a ball or spherical mass. The hearth is so mounted and actuated as to receive an oscillatory motion in opposite directions. The working chamber is constructed on an arc of preferably less than 180 deg., and is provided with openings for the introduction and discharge of material. Two such chambers may be mounted on the same shaft, so that one can be charged or discharged while the other is in operation. The relationship between the curvature of the hearth and the axis of rotation is such that the hearth in action will have a greater pitch than that represented by its curvature. Thus the curve of the hearth may be eccentric to the center of rotation and will give the balled mass considerable travel at each periodic oscillation of the hearth. The flame for heating is introduced at the top of the furnace, so that it may impinge downwardly against the metal or against surfaces which will reflect the heat against the metal. When the furnaces are used in pairs the exhaust from one may be used to minimize the cooling of the other or to preheat the charge in such other furnace. The general object and operation of the furnace will be apparent by reference to the figure without further designation of the parts. (1,029,234-5, June 11, 1912.)

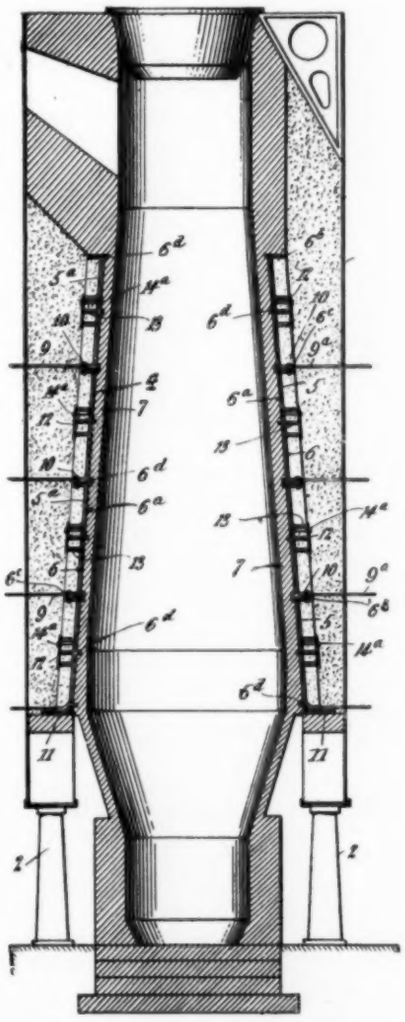


FIG. 6.—BLAST FURNACE CONSTRUCTION.

Blast Furnace Construction.—The reconstruction of the upper part of blast furnaces, whereby a thin artificially cooled wall is substituted for the older style of thick wall, is the subject of recent patents. J. B. LADD and DAVID BAKER, of Philadelphia, Pa., have designed the furnace shown in Fig. 6, which is fitted with water-cooled jackets shown in Fig. 7. The drawing of the furnace represents the usual external shell sup-

ported by columns. In applying the invention, there is erected within the shell and some distance therefrom, a wall 4 consisting of a number of superposed belts or sections 5, 5a, etc., each comprising a series of upright segmental plates. The wall thus formed is lined with fire brick 7, and is supported by the usual columns. In this manner there is provided a continuous circumferential space between the outer shell and inner wall. The plates of which the inner wall is composed are shown in detail in Fig. 6, and will be understood from the drawing. They rest upon each other in the different belts, and are adjacent to each other in the same belt, being held in place by bolts. The cooling is effected by passing water through the pipes contained in the jacket, as shown. The inlet and outlet are shown at 9 and 9a, these openings being connected with suitable pipes to convey water to and from the jackets. Any space left between the outer shell and new inner wall of the furnace may be filled with slag, loam or cement. The primary object of the invention is to substitute a thin wall for the usual massive wall of the furnace, and is especially designed to replace the latter when it has been rendered unfit for further use. By such replacement a great saving in expense is effected in the restoration of the damaged furnace. (1,027,738, May 28, 1912.)

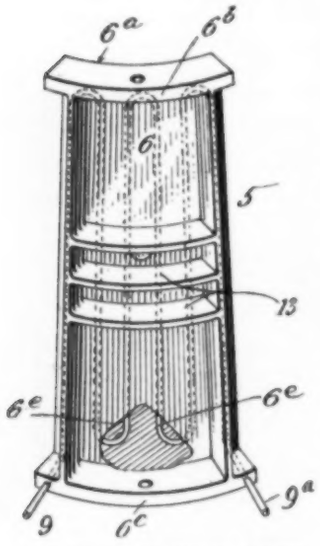


FIG. 7.—WATER-COOLED JACKET.

Another type of construction designed to effect the same result is shown in Fig. 8, wherein is shown the application of water sprays to the outside of the inner wall of a furnace. This design embodies the invention of Max McMurray, of Cleveland, Ohio. The outer shell of the furnace is represented at *A*, resting on the columns *BB*. The lining of the old furnace is completely removed, and there is substituted therefor an inner shell *C*, also of steel plates, which is provided with a comparatively thin lining *C'*, which extends down to the bosh. In the space between the two shells is provided a system of piping for the purpose of spraying the outside of the shell *C*. This consists of a series of perforated pipes *FF* connected with a suitable source of water supply. In order to have access to the spraying system a series of manholes can be arranged which will be accessible from platforms *H* secured to outer shell *A*. An annular trough can be provided near the bottom of the furnace to collect the water flowing down the sides of the furnace. By means of this construction an old furnace can be fitted with a thin, cooled lining without the necessity of removing or disturbing the hoist bridge or furnace top, which are supported by the old shell. (1,024,801, April 30, 1912.)

Sintering Fine Ores.—With the object of agglomerating fine ore, flue dust, mill scale and the like, Mr. James G. West, Jr., of Wilkinsburg, Pa., has patented a process differing from others which have been proposed for the same purpose. This apparatus consists of a furnace of the reverberatory type, having gas burners suitably placed in its sides. The hearth on which the sinter is to be made is movably mounted on wheels and adapted to be run into and out of the furnace on tracks. The bed of the hearth is of refractory material, and a layer of ore or dust is placed thereon before it is run into the furnace. As the agglomerating of this mass proceeds additional charges of the same material are fed through openings in the top of the furnace and spread over the area of the hearth by means of poles or rabbles inserted through the sides of the furnace. When the sintered mass has been built up to the desired thickness the hearth is removed and dumped, the sinter being broken into pieces of suitable size while it is yet hot. The method may prove desirable in special cases, but does not appear to offer the advantages of the more common continuous types which have proved so successful. (1,031,153, July 2, 1912.)

Gold and Silver.

Continuous Vacuum Filter.—The principal forms of filter used in treating ore slime in the cyanide process are those in which a cake of slime is formed on a thin leaf of varying area or on the surface of a revolving drum. In Fig. 9 on page 489 is shown a horizontal type of vacuum filter, designed and patented by Mr. John E. Rothwell, of Denver, Colo. It consists of a series of filters arranged radially in circular or polygonal form, each section having suitable connections with sources of vacuum and pressure. One of the sections is shown in the position assumed when the filtered slime is dumped.

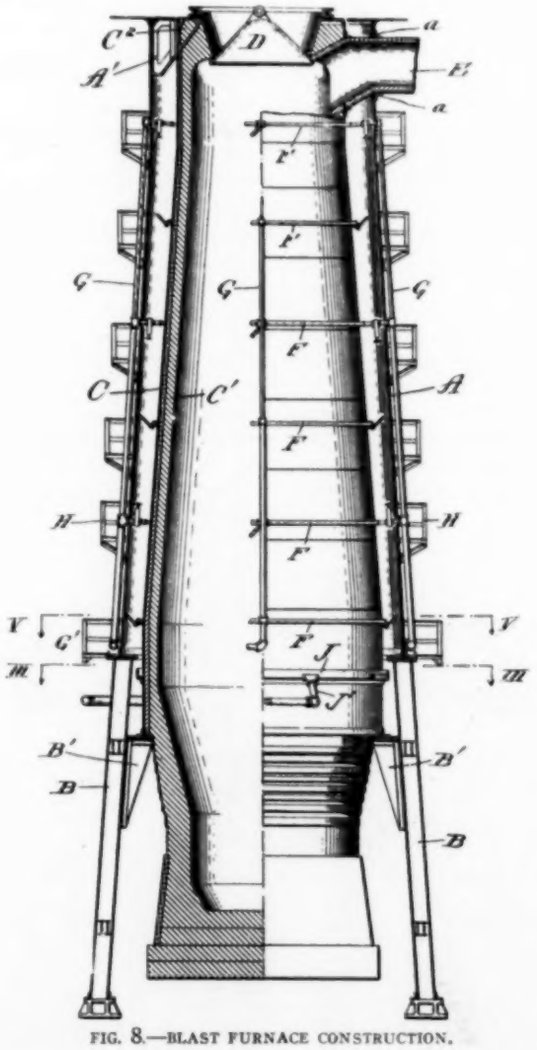


FIG. 8.—BLAST FURNACE CONSTRUCTION.

In operation the entire filter table is rotated by means of power applied through gearing at 9 and 9a. Slime-pulp is delivered to the filter through the feed-launder 2, from which it falls in an evenly distributed stream on to each section as it passes that point. The centrally-located valve is so designed that vacuum is applied to each section as it is loaded with pulp, whereby the valuable solution is withdrawn to a suitable receptacle. In the course of revolution wash solution may be applied at the point *x* and wash water at the point *y* for the

purpose of displacing as completely as possible the valuable solution retained by the slime. These solutions may be withdrawn into separate receptacles, and will not necessarily be

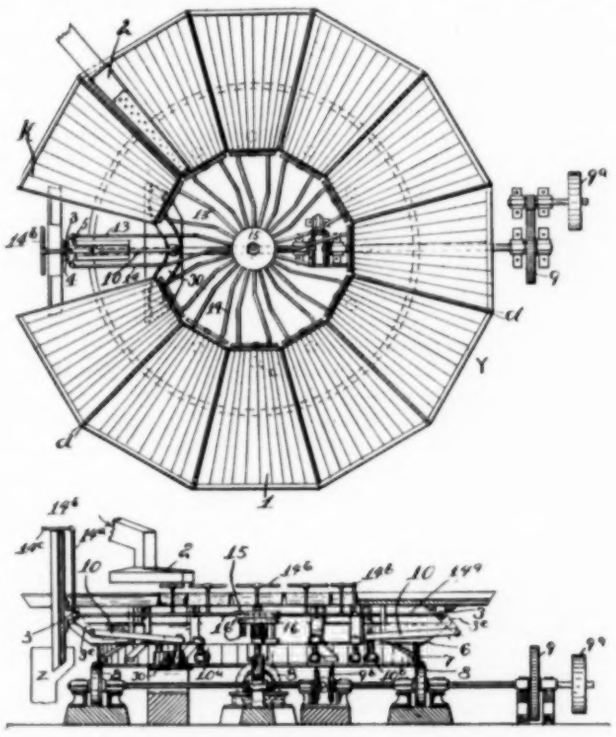


FIG. 9.—HORIZONTAL VACUUM FILTER.

mixed with the first rich solution. On arriving at the dumping point a roller on the end of the lever arm 10 becomes engaged by the curved track 30, which causes the section to be tilted outwardly, as shown. At the same time the central valve causes compressed air to be delivered beneath the filtering medium and the cake is blown off the cloth. For the further cleansing of the filtering surface a stream of water may be applied through the pipe 14^b and caused to play over the vertically tilted section. After the cake is dumped the section passes out of engagement with the curved track and assumed a horizontal position, ready to receive another charge of slime. The entire filter table is supported by a circular tire traveling on the wheels 8. (1,028,789, June 4, 1912.)

Another form of vacuum filter is shown in Fig. 10. It is a modification of one described in these columns December, 1911, page 660, and is the invention of BRUCE W. TRAYLOR, of

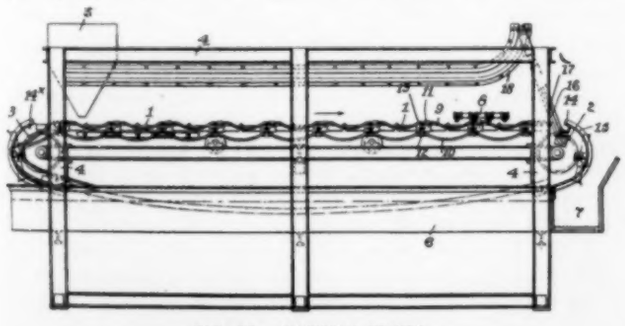


FIG. 10.—VACUUM FILTER.

New York, N. Y. It is composed of rectangular trays 1, arranged in the form of an endless chain. Flexible tubes 9 and 10 provide connections for each tray with the pipes 16 and 17, which, in turn have flexible connections with sources of vacuum and pressure. The valves which provide for alternating connection of each tray with vacuum and pressure, are shown as star-wheels 13. In operation, slime-pulp is delivered to the traveling filter through a feed-hopper 5 and falls on to each tray as it passes beneath the hopper. At this time, and during the period of travel across the filter in the direction of the arrow, vacuum is applied to each filter-tray, and the valuable solution is suitably collected. Wash water can be applied at 8. As each section approaches the turning point at the right, the star-wheel is turned by the guide 14, and pressure is turned on under the filter fabric, thus discharging the cake into the launder 7. On the return trip, the sections travel beneath the tracks and are washed by water contained in a tank 6. On emerging therefrom, the star-wheel is again actuated by the guide 14^a, and vacuum is applied. During the movement of the endless chain of filters, the pipes 16 and 17, which connect with vacuum and pressure, travel back and forth, being guided by the track 18. (1,029,155, June 11, 1912).

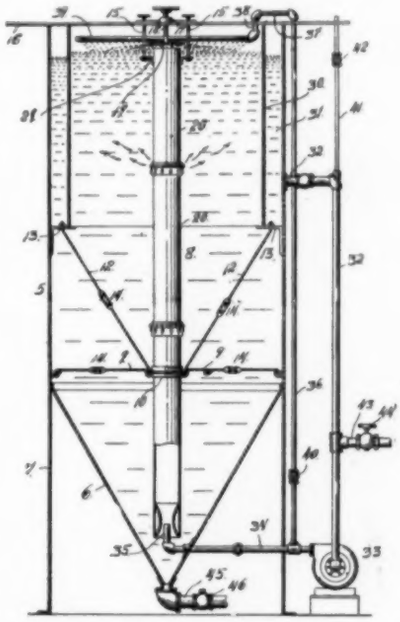


FIG. 11.—PULP AGITATOR.

Pulp Agitator.—Letters patent have been issued to WILLIAM HAMMOND, of Warren, Pa., in the matter of an agitator for ore pulps in cyanide solution. The agitator is illustrated in Fig. 11. It consists of the usual tall, cylindrical tank with cone bottom and central tube. In this case, agitation and circulation are accomplished by means of a centrifugal pump 33 which has the following connections. It receives pulp from the annular space 31 which is formed by the circular apron 30 within the tank. The pulp flows from this space through pipe 32, enters the pump and is discharged through pipe 34 leading to the central tube, with a small quantity passing through valve 40 and pipe 36 to the spray pipe 39 at the top of the tank. At the same time that pulp is drawn into the pump, air may be drawn also through pipe 41, which is an extension of the pipe 32. In this way aeration is accomplished simultaneously with agitation. By diverting a portion of the pumped pulp to the top of the tank, and discharging it through the circular perforated pipe 39, the inventor claims to secure what he terms "cross agitation." This is claimed to result also from the arrangement in the central tube, whereby a portion of the rising pulp can issue from an annular series of holes bored in sleeves surrounding the movable sections of the standpipe, as shown at two points intermediate the top and bottom of the tube. These movable sections can be raised or lowered by means of screws 15. When raised, pulp can issue from the holes in the sleeved surrounding the joint. (1,026,578, May 14, 1912.)

Electrolytic Precipitation of Gold from Cyanide Solutions.—An improved form of electrolytic cell for precipitating the precious metals from solutions obtained in the cyanide or chlorination processes is shown in side and sectional elevations in Figs. 12 and 13 respectively. It is the invention of WILLIAM H. DAVIS, of Boulder, Colo., and will be used in the mill now under construction for the United States Gold Corporation. In other forms of cells for gold precipitation the

electrodes are disposed parallel with the flow of the solution, and as the ionic migration is slow compared with the velocity of flow of solution through the cell, these cells must be of considerable length to insure complete precipitation. It is the

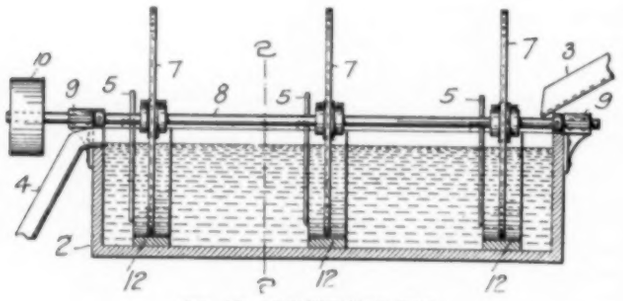


FIG. 12.—ELECTROLYTIC CELL.

object of the present invention to construct a cell in which the flow of solution is an additive quantity to the ionic migration, with the result that time, electric energy and the size of the cell are reduced. This is attained by placing permeable electrodes transversely to the flow of solution, so that every particle of solution must pass through the foramina of the electrode substance, or in such close proximity to the same that the space to be traveled by the ions is reduced to a minimum.

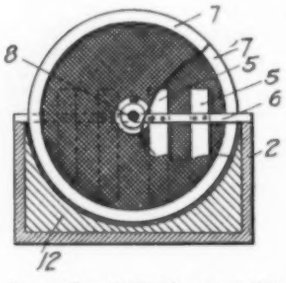


FIG. 13.—SECTION ON LINE 2-2 OF FIG. 5.

In Fig. 12 the cell 2 receives its solution from the launder 3 and discharges it through the launder 4. The anodes 5 are bars suspended from beams extending across the tank, as shown more clearly in Fig. 13. The cathodes 7 are circular, and adapted to be rotated by power applied to the pulley 10. A partition 12 prevents the solution from flowing under the circular cathodes, and forces it to travel through them. A suitable substance of which to construct the cathodes is wire screen. (1,015,546, Jan. 23, 1912.)

Ore Roaster.—The extraction of gold from sulpho-telluride ores by cyanide is facilitated by a preliminary roast. Time and temperature are two elements in the roasting operation which must be properly regulated. Messrs. William H. Davis and E. A. Clark, of Boulder, Colo., have found by experiment and miroscopic observation that a temperature of about 500° C. for a period of two hours is required for proper

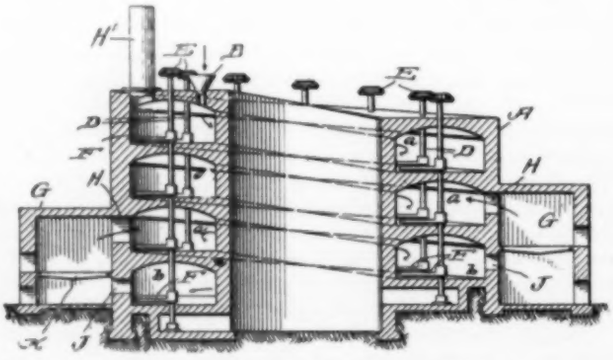


FIG. 14.—HELIXOIDAL ROASTING FURNACE.

roasting. They have designed and patented a type of roasting furnace and cooling hearth, which they term "helixoidal," referring to the structure of its continuous hearth returning to a position vertically below the initial point, whatever the travel of the ore may be. The object is to obtain a form of roaster which will combine a maximum length of hearth with a minimum of floor space.

The form of the roaster is shown in Fig. 14, which is a vertical section. Vertical shafts D are driven from gears E. Attached to these shafts on the hearth are rabbles F, which rotate in opposite directions and effect the mixing and progression of the ore along the hearth. The lower passage *b* is the cooling hearth. Fireboxes G are arranged at suitable places and in number sufficient to provide heat for the roast. The inventors claim economy in operating cost by a greater utilization of the heat units from the fuel. (1,029,667, June 18, 1912.)

Zinc.

Electric Reduction of Zinc.—Among the recent proposals for the utilization of the electric furnace for the reduction of zinc ores is that of Charles V. Thierry, of Paris, France. His proposed furnace is illustrated in section in Fig. 15. His process involves the previous treatment of the ore by fire concentration, whereby an enriched zinc oxide of commercial grade can be obtained and reduced in the furnace with an admixture of carbonaceous fuel. The advantages of this procedure are: (1) The treatment of a product containing a high percentage of zinc; (2) the elimination of those substances which form the slag or residue now obtained in ordinary zinc reduction, and (3) the creation of conditions most favorable for the use of the electric furnace.

Referring to Fig. 15, B represents a furnace retort of suitable construction. A resistor C of particles of carbon is placed on the hearth and suitably connected with a source of current not shown. The ore and carbon mixture is shown at

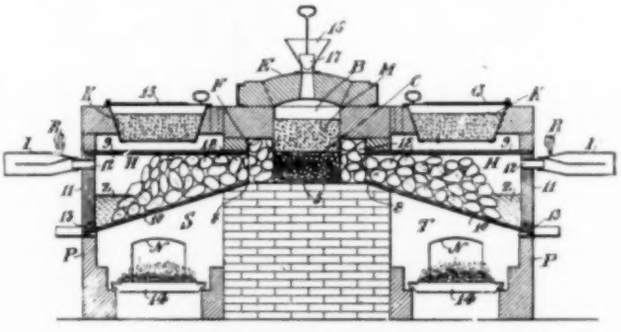


FIG. 15.—ELECTRIC FURNACE FOR REDUCTION OF ZINC.

M resting on the resistor. The retort is provided with a feed-hopper and plug as shown. Openings F from the retort communicate with the condenser H, the latter being filled with coarse coke 18. A tap-hole is shown at 13, and a means for the escape of gas at 12. K represents receptacles for preliminary drying and pre-heating of the charge by utilizing radiated heat. Fireboxes 14 are provided to supply additional heat, if such is necessary to keep the condenser properly warm, or above the melting point of zinc.

In operation, the resistor is brought to the proper temperature, and a charge of ore is placed thereon. As distillation proceeds, the zinc fumes pass through the bed of ore and the resistor into the condenser, where it passes through the carbon therein contained. The metallic fume is thus constantly in contact with reducing substance. The feeding intervals will be indicated by a volt meter, for when a charge is approaching exhaustion the normal resistance will be affected and will serve as an indication that more charge is needed. The determining factor which limits the capacity of the furnace is the condenser. Hence, the construction is such as will give a high rate of production. The inventor states that he has demonstrated the efficiency of this process by running continuously for weeks, producing zinc at the rate of about 75 lb. per hour: that by reason of the comparative purity of the product smelted, most of it being converted into the gases of zinc and carbon monoxide, little slag or residue is formed, and the retort need

not be cleaned save at long intervals. (1,030,349-50, June 25, 1912.)

Treatment of Blue Powder.—The production and retreatment of blue powder formed in zinc reduction has always constituted one of the principal difficulties in the metallurgy of zinc. Mr. CHARLES V. THIERRY, of Paris, France, proposes to treat the blue powder with a dilute solution of hydrochloric acid, in proper proportion to just dissolve the zinc oxide present and leave the zinc in a condition to be dried and melted. He states that he has confirmed his belief that the proper quantity of acid will dissolve merely the film of oxide on the surface of each particle of metal, and leave it in a condition to be melted. The process is designed to be cheaper than resmelting, and to recover the zinc more rapidly. (1,030,351, June 25, 1912.)

Alloys.

Casting Magnesium Alloys.—The difficulties in casting alloys containing magnesium arise from the affinity of that metal for oxygen. BERTHOLD HOFFMAN and ROBERT SUCHY, of Griesheim, Germany, have discovered that this reaction is mitigated by the addition of a small quantity of calcium to the alloy. Alloys thus formed may be cast without any oxide or nitride of magnesium being formed, or any other injurious action taking place. Thus, for instance, magnesium and its alloys, when mixed with from 0.1 per cent to 0.5 per cent of calcium, flow in a silver-white stream without burning taking place or a dark skin forming on the surface of the metal. The castings fill the molds in a perfect manner, and do not effloresce in the air. The property given by the calcium is not lost on repeated melting and casting. The presence of the calcium further makes possible the casting of magnesium or its alloys free from hydrogen bubbles. It is not necessary to use calcium in the metallic form, as it can be applied as powdered burned lime. It may be added also during the electrolytic manufacture of magnesium. Being used in such small amounts, it does not impart to the alloy any of the characteristics of a calcium alloy. (1,028,216, June 4, 1912.)

Synopsis of Metallurgical and Chemical Literature.

Gold and Silver.

Parks Electrocyanide Process.—Among the several proposals for the use of electricity in connection with cyanidation is that of MR. JOHN R. PARKS, of Hollywood, Calif., who describes his system in the July, 1912, *Quarterly* of the Columbia School of Mines. The system is intermittent and yields the precious metals in the form of an amalgam in a single operation, thus eliminating filtering and precipitating devices.

The electrocyanide vat consists of a wooden tub, 15 ft. diameter and 30 in deep, through the center of which projects a small conical casting for the passage of the working shafts. Practically the entire bottom of the pan is covered by an amalgamated copper plate, to which quicksilver is added in proportion demand by the ore. This plate is the cathode. About nine inches above the cathode is a cast-iron anode plate equal in area to the cathode, mechanically suspended in a suitable manner. The anode plate is divided into two annular parts, the outer one containing 60 per cent. and the inner one 40 per cent. of the anode area. These annular anodes revolve in the same direction, and independently, by means of pulleys, gears and a solid and hollow shaft. Wooden paddles are fastened on the underside of the anode, extending to within 2 in. of the cathode plate. The compound rotary motion of the anodes and their paddles gives a swirling motion to the ore pulp, causing it to travel toward the periphery of the vat where it passes upward between the anode and side of the vat, and thence back to the center. Air jets at the periphery of the tank supply oxygen to the solution.

Current is supplied by a 10 to 20-volt, 400-ampere D. C. generator. The positive wire, in four branches, is brought into contact with collecting rings on the shafts which rotate the anodes, and the current is thus distributed to the anode plates. The negative wire is similarly connected to the four sections of the cathode.

In operation, the ore is treated with a cyanide solution of suitable strength, with protective alkali, and a quantity of salt. Complex chemical reactions occur when the current is applied, chief among which is the decomposition of the salt. The sodium liberated is constantly precipitated on the cathode, forming sodium amalgam; the sodium is converted into sodium hydroxide which reacts with the acid constituents of the ore and prevents excessive consumption of cyanide. The cyanide is decomposed also, but there is no question of the subsequent restoration of cyanide, as the author has frequently found more free cyanide of potassium or sodium in a sample of solution taken an hour or so after a previous sample. Mercury should not be lost in the process, but in practice a small loss is observed; in one case this amounted to about 5 lb. in treating 2500 tons ore. Base metals are precipitated as well as the gold and silver, and base bullion can be produced.

The power required for a single vat is 0.6 hp for mechanical agitation, 1 to 1.5 hp for compressed air, and 4.5 to 6 hp for electric current. A single charge, of 1.1 pulp, is 7½ tons, and the time of treatment varies from 2½ to 4 hours. The daily capacity of a vat, including time taken for discharging, is from 35 to 60 tons.

The author appends a few examples of results obtained on different ores, which show recoveries of from 86 to 97 per cent. of the gold and from 62 to 91 per cent. of the silver, as amalgam.

Electric Melting of Gold Precipitate, and Smelting of Concentrate.—The Lluvia de Oro Gold Mining Co., Lluvia de Oro, Chihuahua, Mexico, uses electric resistance furnaces for melting the gold and silver precipitate from cyanide solution, and for smelting roasted gold and silver concentrate. The work is described by H. R. CONKLIN, manager for the company, in the *Eng. & Min. Jour.*, June 15, 1912.

The four furnaces are rectangular in section and have the following inside dimensions: 16 in. square and 4 ft. deep. The side and back walls are of solid adobe. All bricks, except the front course, are liners. Chrome dust, with sodium silicate as a binder, was first used for lining, but lasted only about thirty hours. Good firebrick, laid with ends exposed to the charge, and having the joints floated full of fireclay, last about 200 hours. In an effort to improve this, one lining was tried of carborundum firesand, rammed 3 in. thick, with 5 per cent sodium silicate of 52 deg. B. for a binder. This was too good a conductor and reduced the capacity of the furnace. The life was about the same as for firebrick, but cost twice as much. For a tap-hole, the brick lining in front is built with a tapered channel leading from the furnace for 3 in. or 4 in. to an iron plate with a ⅞-in. hole. A spout of angle-iron is laid from the plate to guide the flowing metal to the molds.

Alternating current of 60 cycles is used at 110 volts. Acheson graphite electrodes have given the best service. On account of revolutionary troubles in 1911 iron electrodes were used for a time, but they are objectionable in smelting concentrate on account of adding iron to the metal and causing difficulties in discharging the furnace.

Fluxing of the precipitate is varied according to its character. A customary charge is 100 parts precipitate, 5 of lime, 15 of sand, 10 of borax and 10 of soda. In starting the furnace all resistance is cut into the circuit and the upper electrode lowered to the bottom of the furnace in one corner. A handful of powdered graphite is thrown in between the electrodes, followed by borax and powdered slag until the bottom of the furnace is covered with melted slag. As slag accumulates the upper electrode is raised. The resistance is gradually cut out, and is usually all out in thirty to fifty minutes, with a pool of

molten slag in the furnace about 1 ft. deep. Fluxed precipitate is then added until the furnace is full, and the resulting bullion cast into molds. The bullion runs about 900 fine.

Short zinc is roasted, fluxed and melted. The resulting bullion is remelted with precipitate.

Concentrate at this mine is worth about $1,500 per ton. Formerly it was sacked and shipped to the smelter, but it is now reconcentrated and smelted electrically after roasting. The cost of marketing concentrate was about 5 per cent higher than the cost of marketing the same value of bullion. The present process consists in reconcentrating to a value of about $3,000 per ton, roasting off nearly all the sulphur in a small reverberatory furnace, fluxing and smelting in the electric furnace. A charge contains 100 parts concentrate, 15 of lime, 30 of sand, and 15 of slag from precipitate melting. The products are bullion and a high-iron slag.

Only bullion is shipped from the mine. Slag from precipitate is used as flux for concentrate, or sent back to the mill for treatment by cyanidation. Slag from short zinc is sent back to the mill. Slag from concentrate is stored until a sufficient quantity has accumulated to warrant separate treatment in the mill. Furnace refuse, bricks, clay, sweepings, etc., are sent to the mill. It should be added that the company owns its hydro-electric power plant, the operating cost of which is constant whether the output is a maximum or a minimum.

Regeneration of Cyanide Solutions.—Cyanide solutions which become foul in treating cupriferous ore may be regenerated, and the copper recovered, according to a method practiced by Mr. W. D. Williamson, who has communicated his ideas to the *Journal* of the Chem. Met and Min. Soc. of S. Africa, April, 1912. The addition of sulphuric acid to such a solution will cause the following reaction:

$$Cu_2(CN)_2 \ 2KCN + H_2SO_4 = Cu_2(CN)_2 + K_2SO_4 + 2HCN.$$

The insoluble copper cyanide can be separated by filtration, and the filtrate neutralized by a slight excess of alkali. It is obvious, however, that there is still cyanogen in the copper precipitate which ought to be recovered if possible. If the precipitate be suspended in water, and treated with hydrogen sulphide the following reaction occurs:

$$Cu_2(CN)_2 + H_2S = Cu_2S + 2HCN.$$

The solution resulting from filtration of the mixture can then be neutralized with alkali and used as solvent for fresh ore.

Finally the author combined the two reactions by adding sodium sulphide and the requisite amount of acid to the cupriferous cyanide solution, with this reaction:

$$2KCN \cdot Cu_2(CN)_2 + Na_2S + 2H_2SO_4 = Cu_2S + 4HCN + Na_2SO_4 - K_2SO_4.$$

The precipitated copper cyanide carries down with the precious metals and can be collected, dried and shipped to the smelter. The clear liquor is then neutralized with a slight excess of alkali to combine with the free HCN.

Synthetic Rubber.

Prof. Wm. H. Perkin's paper, recently presented before the Society of Chemical Industry in England on "the production and polymerization of isoprene and its homologues" has resulted in a great deal of newspaper articles and also some discussions in the British technical and especially electrical journals, in view of the importance of rubber for the electrical industry.

The London *Electrician* of June 28 writes as follows: "The first step in the process is the production of fusel oil and acetone from starch-containing substances by fermentation. The next operation consists in subjecting isoamyl alcohol, which is obtained by distillation from the fermentation products, to the action of hydrochloric acid, while in the third the vapor of the isoamyl mono-chloride thus obtained is treated with chlorine gas, resulting in the production of isoamyl dichloride. This latter substance is then passed over lime heated in an iron tube and the isoprene so formed is polymerised into rubber by contact with metallic sodium, the change being assisted by heat and taking place in a few hours, weeks or days according to the temperature employed. In this way it is possible to build up by a true synthetic process a rubber closely resembling the latex that comes from the tree." . . .

The editorial then continues to say: "The company which is being formed to work the processes detailed above will, we are informed, for the present confine itself to the synthetic production of acetone and fusel oil alone, two substances which it will be able to put on the market at prices much lower than those at present existing. This should have a great effect especially in the explosives industry where large supplies of acetone are required. Fusel oil, also, has risen in price very greatly during the last few years. But as regards rubber no steps are at present to be taken, so that cable manufacturers are not yet likely to be able to obtain this commodiy at a low price; while, on the other hand, those of our readers who own shares in rubber companies may sleep quietly in their beds."

The London *Electrical Review*, of June 28, writes as follows:

Prof. Wm. H. Perkin's paper "essentially consisted of an historical review of the attempts to make rubber artificially, followed by an account of the work conducted by an organization of chemists directed by Messrs. Strange and Graham, working in conjunction with Prof. Perkin and later with Prof. Fernbach, of the Pasteur Institute, Paris. Prior to the advent of Prof. Fernbach on the scene Sir William Ramsay had already become associated with the group as consultant. The history of synthetic rubber may be said to have started with the discovery by an Englishman, namely, Greville Williams, in 1860, of the fact that among the products obtained by the dry distillation of rubber was a substance, subsequently termed isoprene, which on standing in a sealed veessel thickened and became viscous. In 1875 Bouchardat, of Paris, discovered that by the action of hydrochloric acid on isoprene (also obtained from rubber), a substance which closely resembled natural rubber could be obtained. Subsequently Prof. Tilden found that isoprene could be derived not only from rubber, but from another natural product—namely, turpentine—by passing this substance through a heated tube under certain conditions, and from the isoprene so prepared Prof. Tilden was able by two processes to obtain a material, the qualities of which appeared to be much the same as those of natural rubber. The next step was to clear up the constitution of the material isoprene, and, to pass over all the intermediary stages, it may be stated that this substance has been proved to be a methyl-butadiene—that is to say, a chain compound containing five carbon atoms and two double linkages. For a time the views of Tilden regarding the basis substances of natural rubber were strenuously combatted by a German school of chemists, headed by Prof. Harries, and the failure of Harries to obtain rubber from isoprene as described by Tilden led him to express a serious doubt doubt as to whether Tilden had not been wrong in his observations, or had not drawn wrong deductions from his experiments.

"Not the least interesting part of Professor Perkin's discourse consisted in the production of some of the original specimens of the rubber-like material prepared by Prof. Tilden, and the refutation of the claims of the chemists of the German school to the effect that they were the original and true discoverers of the synthesis of rubber. That isoprene can be converted into rubber is now acknowledged by the German school, inasmuch as recent researches have enabled them to convert this material into the substance in question."

The London *Electrician* of July 12, contains an article by C. J. Beaver (of W. T. Glover & Company) who emphasizes that "no synthetic rubber appears to have yet been produced which is comparable with high-grade natural rubber as regards physical properties and perhaps at this stage it is hardly fair to expect it."

Mr. Beaver also thinks that lower prices will naturally arise in due time owing to new rubber plantations, thus rendering synthetic rubber less necessary commercially.

The "Clam Shell" Filter.

BY ERNEST J. SWEETLAND.

Since the Sweetland filter press was first introduced there has been an increasing demand for a modification of the principle that would lend itself to conditions where dry discharge of the residual matter is required. True, a number of large Sweetland presses of the frame type have been adapted for dry discharge and are doing the work for which they were designed. These have been constructed with a large door in the lower end of the press, and while the cakes are held in place on the leaves by a partial vacuum, a tray is inserted through the door and the cakes caused to fall. The tray with its load is then removed, the door closed, and the press again charged. In some cases the use of the tray is dispensed with and the residual matter withdrawn with a hoe like ashes from the door of a furnace. Most of the machines of this type have been placed where the bulk of residual matter is comparatively small, that is, where the cakes formed were thin and the weight of solid matter to be withdrawn moderate.

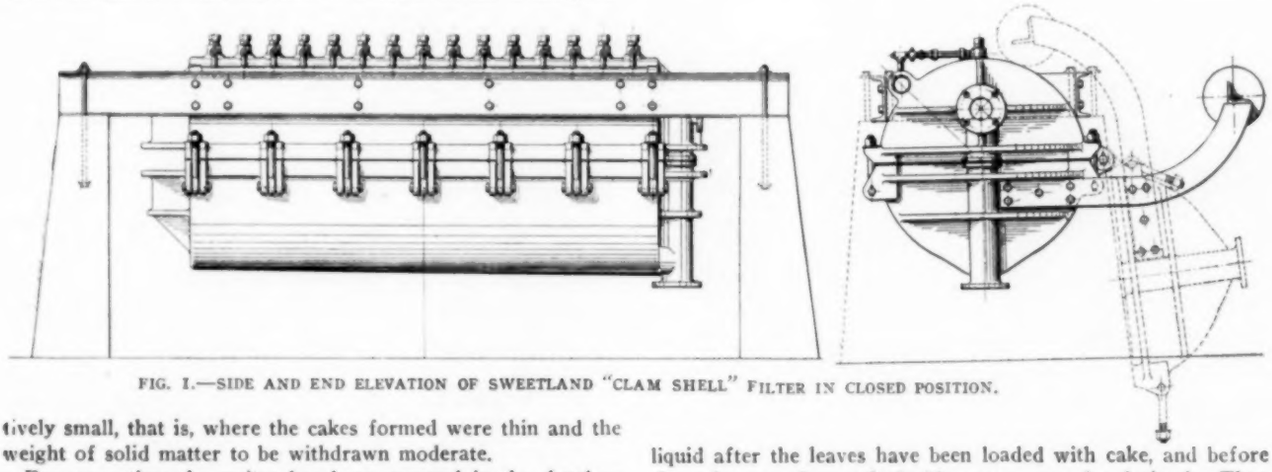


FIG. 1.—SIDE AND END ELEVATION OF SWEETLAND "CLAM SHELL" FILTER IN CLOSED POSITION.

For some time the writer has been engaged in the development of a filter that would accomplish the following important objects, and at the same time not sacrifice the advantages that have been gained in previous types. These objects are:

First.—"Dry"* discharge of the cakes without handling (mechanical or otherwise) and without moving the filter leaves.

Second.—To overcome the objection of having a quantity of excess or surplus unfiltered liquid to handle with each operation.

*The word "dry" as used here is not to be taken in its literal sense, but refers to the filter press cake discharged from the filter in its natural condition, without being disintegrated and flushed out with water, as is the case with the sluicing type.

It is an important point in many processes to have the filter press cake as dry as possible. For example, if precipitated carbonate of lime is to be recovered from its solutions to be reburned, each unit of moisture that remains in the cake must be evaporated. Therefore the drier the cake is discharged from the filter the less fuel will be required. For such a purpose a sluicing type filter would be useless, for the act of disintegrating the cake with water in discharging would defeat the purpose for which the filter is required.

The excess unfiltered liquid referred to under the second head is that liquid which lies in the filter chamber between and below the cakes.

To better illustrate this point, suppose a filter chamber to contain a series of filter leaves located in the upper portion. If a space is left in the bottom of the chamber to make room for a discharge tray, conveyor, or otherwise to facilitate discharging, this unoccupied space contains surplus unfiltered liquid after the leaves have been loaded with cake, and before the cakes can be washed this space must be drained. Then when washing has been finished this space contains surplus water which must be withdrawn before a "dry" discharge could be attempted.

This surplus or excess liquid is the cause of more complications than are visible on the surface, not the least important of which is the enrichment of the wash water. It is not ordinarily permissible to use fresh wash water each charge and discard the excess remaining in the chamber; therefore the excess must be used over and over. Each time the excess comes in contact with a batch of cakes pregnant with a strong solution, a certain amount of the strong solution mixes with the wash water, gradually enriching it until eventually this

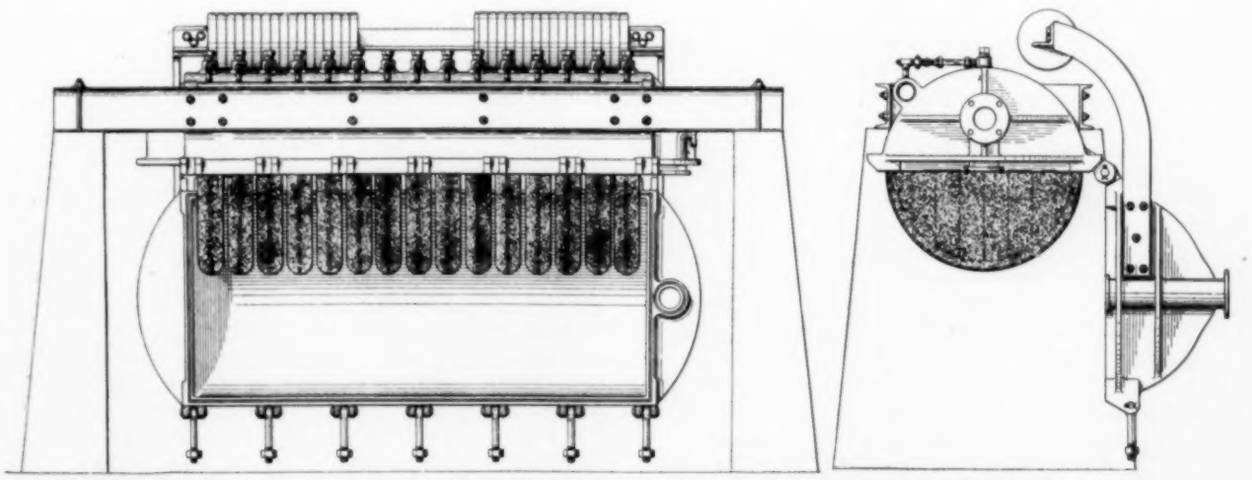


FIG. 2.—SIDE AND END ELEVATION OF SWEETLAND "CLAM SHELL" FILTER, OPEN, READY FOR DISCHARGE OF CAKES.

wash water becomes weak solution and the efficiency of the wash is impaired. Suppose, for instance, the cakes contain a strong solution of caustic soda, and it is desired to displace this solution in the cakes with water. If pure water is available for washing it is possible to completely remove the caustic soda from the cakes; but if the wash water contains caustic soda in solution, then washing ceases to be effective when the effluent is reduced to the strength of the ingoing wash water.

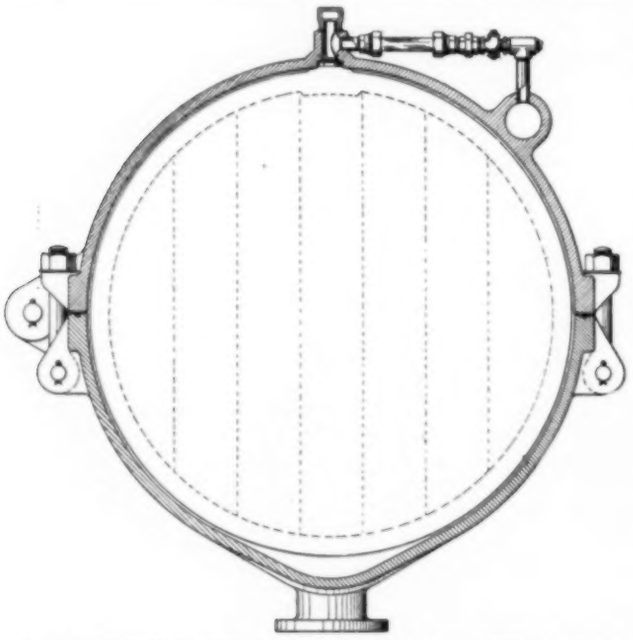


FIG. 3.—SECTIONAL VIEW OF SWEETLAND "CLAM SHELL" FILTER, SHOWING GLASS DELIVERY TUBE AND FILTER LEAF CONNECTIONS.

When this point is reached caustic soda is being washed *into* the cake exactly as fast as it is being washed out. The same applies to an even greater extent in the cyanide process. In this process it is important to have wash water free from gold or silver in solution, as otherwise an appreciable amount of recoverable values are lost with each discharge.

In addition to this, simplicity and convenience in operation are promoted by doing away with excess solutions. Accessory equipment is minimized and the possibility of an inexperienced operator allowing the cakes to fall while withdrawing the excess sludge is removed, the time required for each cycle is reduced and the daily capacity of the installation per square foot of filter area is increased. When there is no excess solution to deal with, the end of filling the press and beginning of washing is marked by the attendant simply turning a three-way valve on the suction of the feed pump, which cuts off the sludge and connects the wash water.

A comparison of the number of steps required for each operation with and without excess solution illustrates the point:

WITH EXCESS.	WITHOUT EXCESS.
Form cakes.	Form cakes.
Withdraw excess.	Wash cakes.
Fill with wash water.	Dry with compressed air.
Wash cakes.	Discharge.
Withdraw excess wash water.	
Dry with compressed air.	
Discharge.	

The above shows that by eliminating the excess the number of steps is reduced from seven to four. Furthermore, it is usually necessary to have a vacuum pump and receiver for use in holding the cakes to the leaves during the transfer of excess solutions, and this equipment is unnecessary when excess solutions are eliminated.

To eliminate excess, or surplus solutions, the "dead" space in the filter chamber must be done away with, that is, when the filter is charged the cakes and the leaves which support them should fill the entire chamber, with the exception of a narrow slit between adjacent cakes to permit the wash water to distribute itself. There should be no pockets to retain unfiltered solution.

The cylindrical form chamber with circular leaves equally spaced and disposed at right angles to the axis of the cylinder makes the accomplishment of the above objects possible. The cylindrical form is of especial advantage because it gives the maximum strength for a given weight, exposes the leaves to view when opened, and permits of easy discharge of residues.

The Sweetland "clam shell" construction reduces void space to such an extent that surplus solutions are reduced to a negligible quantity. The body consists of two semi-cylindrical shells hinged together. The upper member is supported in a stationary position upon the foundations. The lower member is balanced on its hinges by counterweights so that it will swing freely and remain in any desired position. The filter leaves are held in the upper member by a lock-nut on the outside of each; they have no connection whatever with the lower member. (See Fig. 3.)

The construction of the filter leaf varies widely, and must be selected to suit the material in hand. A simple form of leaf is made with a circular supporting frame of flattened seamless steel tubing, perforated on the inner periphery. The ends are united in a casting which receives the outlet nipple. The outlet nipple has a side opening which aligns with the delivery pipe. This frame carries a web of two layers of screen which provides the necessary drainage for filtrate. The whole is surrounded by a bag of suitable filter cloth or duck through which the outlet nipple passes. A locknut on this nipple screws down on the cloth, preventing leakage where the nipple passes through. The delivery pipe carries a shut-off cock and section of gauge glass which renders the delivery from each leaf visible and controllable. Any leaf may be cut off without interference with the rest. The filtrate from all leaves is delivered into a general conduit on the side of the stationary member. Any leaf may be removed from the open filter by removing the locknut at the top.

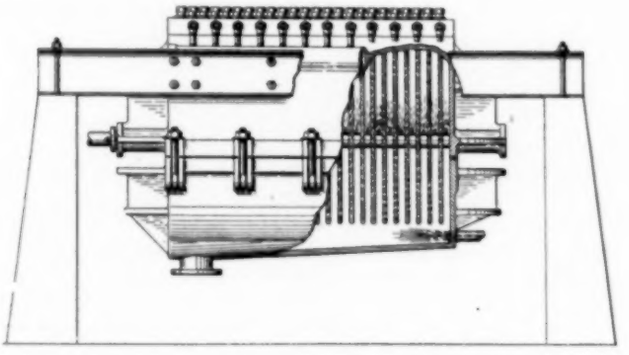


FIG. 4.—SIDE ELEVATION OF SWEETLAND "CLAM SHELL" FILTER, SLUICING TYPE, SHOWING CENTRAL ROTATABLE FLUSHING PIPE FOR WASHING LEAVES AND "BOOSTER" NOZZLE IN THE BOTTOM.

Limited space makes it impossible to go into details regarding the mechanical closing and locking device employed on large units. The moderate-size units are held closed by swing bolts as shown. In all cases the lower member carries an inserted rubber gasket which abuts against the smooth surface of the upper member, forming a water-tight joint. The operation of the filter is as follows:

The liquid to be filtered is forced into the filter body either by pump or monte-jus through an inlet in the stationary member, a slight depression in the bottom of the lower member forming a gutter which permits the liquid to flow along the bot-

tom and enter freely between the leaves. The entire body of the filter is filled before the air vent at the top is closed. The pressure of liquid bears equally against the sides of all the filter leaves and causes the liquid content to pass through the filter cloth while the solid matter deposits on the surface, forming cakes. A testing device enables the attendant to gauge the thickness of the cake at any time. When they have arrived at such a thickness that there is but a small space between adjacent cakes, the supply of sludge is shut off and wash water turned on simultaneously. This change is made so quickly that the pressure in the filter chamber is steadily maintained.

The wash water first displaces the small quantity of liquid between the cakes, depositing upon the cakes the solid matter which it contained. The wash water continues to percolate through the cakes until the solution in the cakes is almost, or entirely, displaced. The washing completed, compressed air is

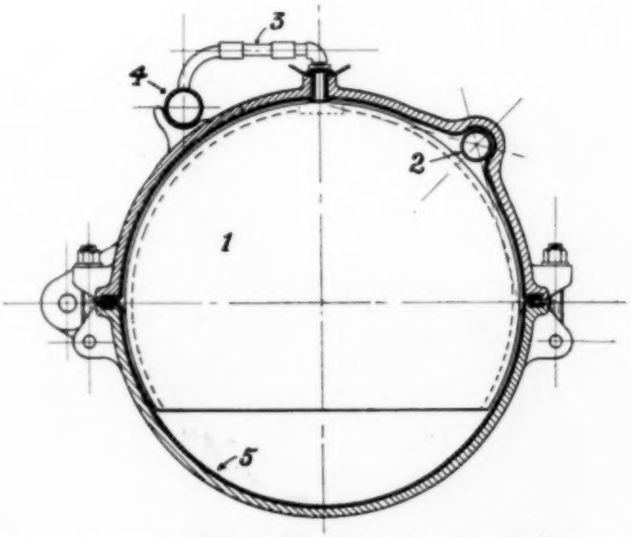


FIG. 5.—SECTIONAL VIEW OF SWEETLAND "CLAM SHELL" FILTER, LEAD-LINED SLUICING TYPE FOR ACID SOLUTIONS.

1, Filter leaf. 2, Flushing pipe. 3, Glass tube. 4, Lead pipe. 5, Lead lining.

admitted, which drives the small amount of liquid remaining out through the cakes, leaving them compact and free from excess moisture.

The final operation is discharging. This is done by unlocking and swinging the lower portion of the shell open and admitting compressed air to the interior of the leaves. The sudden rush of air from the interior of the leaves outward dislodges the cakes, causing them to fall into a truck or conveyor below. If desired a moderate vacuum may be maintained in the leaves while the lower half of the shell is swung open to prevent the possibility of the cakes falling prematurely.

That class of filtration sometimes designated as clarification demands different treatment. Under this head comes the filtration of liquids containing a very small percentage of solid matter, and usually the recovery of the clear filtrate is the desired object, the solid content being refuse to be disposed of by the quickest and cheapest means. In such cases the general construction described is retained, and a small space is left below the leaves, and a rotatable flushing pipe having a nozzle adapted to project a stream along the surface of each leaf passes through the shell from end to end. In this type the adhering slime is flushed from the leaves almost instantly by the streams from the nozzles sweeping forcibly over the surfaces of the leaves as the pipe is rotated. The cleansing action may be assisted by admitting air under moderate pressure to the interior of the leaves. A "booster" nozzle at one end sweeps the fallen slime out of the space below the leaves and through the discharge valve on the bottom. Thus the body of the filter and the filter cloths are quickly cleaned while in normal working position.

A cross-section of a lead-lined filter of this type for acid solutions, constructed with the sluicing pipe at the side of the leaves, is shown in Fig. 5.

This filter and various modifications are fully protected by patents and pending applications owned by the Sweetland Filter Press Company, New York.

An Electric Furnace for the Treatment of Tin Dross, Concentrates from Cyanide Mills and Other Metallurgical Work.

By R. S. WILE.

The old and still the most prevalent way of smelting tin drosses is in the reverberatory furnace, which causes varying but large losses of tin owing to the nature of the furnace.

FIG. 1.—ELECTRIC FURNACE IN OPERATION.

The average loss is in the neighborhood of 8 per cent and then it is necessary to add lead in order to keep the loss so low. This of course makes the metal unfit for straight tinning and it is used for terne plating instead.

In 1908 we tried some experiments in reducing this class of material in the electric furnace and very early hit upon the plan which we have followed in most all of our types of furnaces since then. This dross contained from 36 to 56 per cent tin, 9 to 15 per cent zinc, and 10 to 18 per cent iron, with alumina and other impurities included. The dross was a by-product of tinplate mills and was the residue obtained after all the metallic tin, which was gathered with it in skimming the tin pots, had been melted out. Because of the zinc, lime and iron, it was extremely difficult to recover the tin in a comparatively pure state without great loss, and that this loss did occur is readily comprehended from the foregoing.

In the style of furnace we adopted, we decided to melt the slag first. This is not feasible in a reverberatory furnace, because it will chill when the cold dross is thrown in and it is then almost impossible to remelt it. After this slag was melted in the electric furnace, which was of the shaft type (the slag having a depth of three to four feet, depending upon

the size of the furnace, and in any event taking up half the height of the furnace), the dross mixed with the proper amount of carbon and with enough silica to flux the impurities was thrown in on top and the furnace completely filled. The dross stays on top of the slag and the molten tin trickles through it and is tapped from the bottom. There is no volatilization, as all fumes are condensed in the colder portions of the dross, the only gases escaping being those caused by the reduction, which bubble up through the dross without carrying any metallic fumes with them.

We found out some very interesting things in connection with the smelting. Of course, we knew we could cause a quicker reduction by using more current, but one of the things we learned was that by increasing the temperature of the furnace we decreased the loss of metal. By employing pyrometers we found that we, too, could have a loss of 8 per

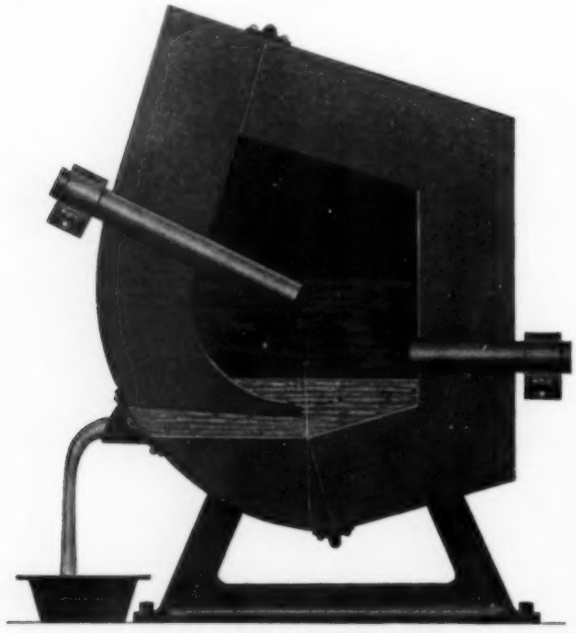


FIG. 2.—ELECTRIC FURNACE TILTED.

cent by running the furnace at 3000 to 3600 degrees Fahrenheit, but by increasing this to 4200 degrees the loss was in the neighborhood of one-half of 1 per cent. As a consequence we ran the furnace at the higher temperature, using more current, of course, but obtaining more metal to offset this and very materially reducing the loss. Slightly better results were obtained from direct current than from alternating, not only because there were no induction losses, but because the direction of the current could be changed at will, making the top or bottom positive as desired and consequently, hotter at the point of entry. It might be well to state that in using alternating current we get a power factor of at least 93 per cent, because the voltage employed is rather high for furnace work, varying between 110 and 220.

While working a set of these furnaces, the writer was visited by Mr. H. R. Conklin, of the Lluvia de Oro Gold Mining Company, of Lluvia de Oro, Chihuahua, Mexico, who thought our type of furnace would be adaptable to the melting of the finely divided gold obtained from cyanide plants. He ordered a set of these furnaces for this purpose and aside from some slight changes in regard to size and material of construction, these furnaces have been working ever since then, and in fact have been increased in number from two to four. These furnaces are smelting their concentrates as well as melting the zinc precipitates.

We have done a great amount of experimenting on ore and concentrate smelting and find that this type of furnace is not only feasible in many cases, but absolutely necessary in some. Take, for instance, an ore with a low copper content, high in zinc, and a gangue material of lime and iron garnet. All these have about the same specific gravity and magnetic separation is too expensive because of the low value of the ore. We have found that an ore of this kind can be handled very easily, using about 180 to 200 kilowatts per ton. The zinc and lime in the ore coupled with the low value, make it almost impossible to treat any other way. Apart from this, however, is the saving in operation where fuel is scarce and waterpower is not. It was this item that caused the Lluvia de Oro to change their method of melting their precipitates.

In melting the zinc precipitates the operation is similar to the dross smelting, the slag is very similar and the material added in the same manner, the difference being the absence of carbonaceous material as in the case of drosses. The bullion is tapped out into molds, and the slag, which contains very little values, after being used for a very long time, is either milled or used in concentrate smelting.

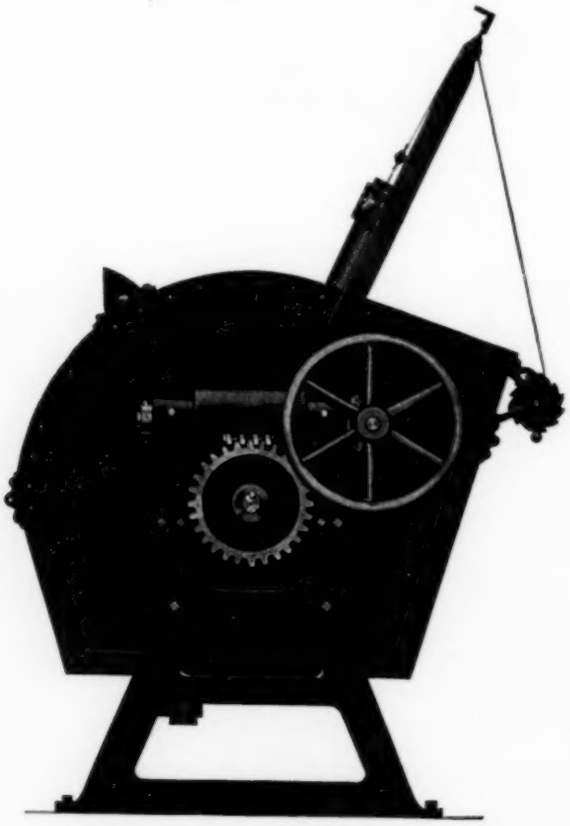


FIG. 3.—OUTSIDE VIEW OF TILTING ELECTRIC FURNACE.

In concentrate smelting where there is a preliminary roast, we have for some time adopted the idea of blowing a jet of air through the taphole to oxidize the remaining sulphur. This has been effective and does away with any sulphides becoming mixed with the bullion. We can do this to a large extent with concentrates running low in sulphur, such as lead-silver concentrates. The slag loss is low in this case also and only about $3.00 value is left after milling.

Ore smelting in the electric furnace is new and conditions are vastly different from those in other furnaces, and while we have tested our furnaces on various types of ores and have worked these out satisfactorily, we are not resting but are working out new ones all the time. Some are not suited to the electric furnace, but a good many are and in the end our work may have been of some assistance in solving vexatious problems.

Pittsburgh Electric Furnace Company,
Pittsburgh, Pa.

Recent Improvements in Uehling Carbon Dioxide Meters

The Uehling Instrument Company, of Passaic, N. J., have recently introduced several novel improvements in their CO_2 recorders and waste meters. To make a CO_2 meter of real value in a boiler room it must be foolproof and must operate without the attention or care of an expert. A great deal of the repairs and attendance required to keep a CO_2 recorder in first-class condition centers around troubles due to the use of a caustic solution and the necessity for its renewal, corrections for errors due to its evaporation or changes in temperature. Thus the principal object of the improvement sought by the Uehling Instrument Company, was to eliminate the caustic solution.

FIG. 1.—VIEW OF ONE BOILER PLANT OF THE CORN PRODUCTS REFINING COMPANY, EDGEWATER, N. J., WITH UEHLING DC WASTE METERS.

The use of any absorbent for CO_2, other than a liquid, would be impossible with any analyzer or CO_2 recorder in which the volumetric displacement of the caustic enters into the measurement of volume of gases. However, as the Uehling CO_2 apparatus does not measure CO_2 by measurement of volume of samples of gas but by a measurement of drop in pressure between two orifices where the CO_2 is removed, it is evident that a dry method of CO_2 absorption would be just as effective as the older or liquid method. Accordingly for some time past investigations have been made in the laboratory of the Uehling Instrument Company covering tests on different types of dry absorbent cartons, with special reference to their adaptability for simple renewal and effectiveness under different operation conditions.

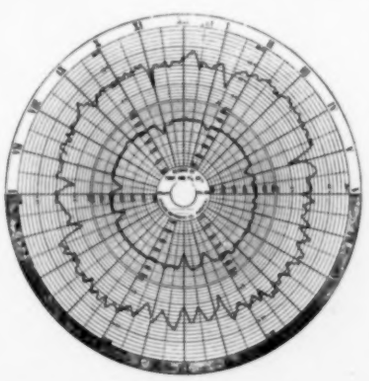


FIG. 3.—THE UEHLING M RECORDER, GIVING A CHART OF CO_2 OR TEMPERATURE, OR IF DESIRED, A COMBINED RECORD ON ONE CHART OF CO_2 AND FLUE GAS TEMPERATURE AS SHOWN IN FIG. 4.

FIG. 4.—CHART FROM UEHLING COMBINED CO_2 AND TEMPERATURE RECORDER.

Experimental work was completed about a year ago, and since then dry absorbers have been installed on practically every CO_2 apparatus out in service, and have been furnished with all new equipments. An installation of this type, which is also provided with the new type M recorder, further described below, is in the boiler plant of the Corn Products Refining Company, at Edgewater, N. J., where all of the twenty-four boilers are equipped with Uehling waste meters for measuring CO_2 and flue gas temperature, one boiler room being illustrated in Fig. 1.

The dry absorber cartons are cylindrical in shape, about 3 in. in diameter and 1½ ft. long, encased in a wax press board, so that they are handled, shipped and stored like ordinary dry batteries, for instance. The absorbent is a specially manufactured flaky substance named natron, carefully packed in layers within the carton. The carton is placed in the cylindrical chamber at the right of the machine, shown in Fig. 2, which illustrates the style S C machine, the operation consisting simply in removing a few wing nuts, removing the used carton and inserting the new one.

Other improvements in connection with Uehling CO_2 meters center around the recording devices proper. The Uehling CO_2 equipment consists of the machine in which the CO_2 is absorbed and the percentage of CO_2 measured pneumatically by changes in vacuum between two orifices, and then this vacuum is transferred to recorders or indicators situated where desired. The new type M recorder is shown in Fig. 3. It is for service in connection with CO_2 meters or the pneumatic pyrometer (operating on a similar principle) or the waste meter which measures both CO_2 and temperature. The type M recorder is simply a light pressure or vacuum recorder for vacuum or pressures, or both, from 0 in. to 60 in. of water column. When this is supplied for recording CO_2, the chart is calibrated from 0 to 20 per cent CO_2, and when supplied for recording temperature it is calibrated up to 1000 deg. or for other ranges as desired. This recorder supersedes a somewhat larger gage used in the past, but operated on the same principle, viz.: the U-tube principle being filled with a

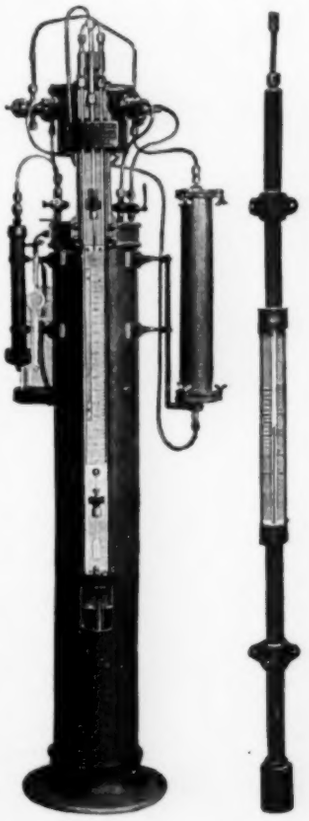


FIG. 2.—UEHLING SC WASTE METER WHICH IS A COMBINED CO_2 MACHINE AND FLUE GAS PYROMETER.

FIG. 5.—UEHLING CO_2 INDICATOR FOR LOCATION AT THE BOILER FRONT.

special oil and involving no levers, pins, springs or similar devices.

In connection with Uehling waste meters, the CO_2 and temperature may be recorded on separate type M instruments properly calibrated, or else a new instrument just developed may be used and the temperature and CO_2 recorded on the same chart. This instrument, the type D M, is built on the same principle as the type M, but carries two pens and gives a chart as shown in Fig. 4.

Fuel Economy in Zinc Metallurgy, Power Production and Mining

By Joseph A. Dunn.

The problem of securing the greatest amount of energy from the fuel consumed for manufacturing purposes, or, in other words, burning everything burnable in the fuel and applying every unit of heat to some direct and definite end, whether that end be the heating of water for steam power, the roasting or smelting or ore, or some kindred purpose, has occupied the attention of engineers and fuel users for many decades. The manufacturer who got the greatest returns from the fuel consumed in the shape of heat energy had the cheapest power. In some instances the question of cheap power has meant success or failure to the manufacturer.

The cheapest power obtainable has been held to be water-power and next to that, in the minds of most manufacturers and engineers, has been natural gas. That ordinary coal can be burned cheaper than natural gas, even when the user of the gas can secure it at the rate of 3 cents per 1000 cu. ft. of gas, is a surprise to most users of fuel, but that this can be done has been successfully demonstrated in the gas fields of Pittsburg, Kan., by the International Engineering Corporation, of Denver, Col., which has installed several of the Parkison patent furnaces, owned by the corporation, under zinc ore roasters and smelters in that section.

A brief account of the furnace and the company will be given first and then concrete instances of commercial users of the furnace, with their opinions of it.

The Parkison furnace, patented and owned by the International Engineering Corporation, was invented by Colorado men to meet a special need in the mining regions of that state. Cheap lignite coal was plentiful near the Colorado Springs and Cripple Creek regions, while the higher-priced bituminous coal was shipped from a distance. Mine owners and engineers were seeking a means of burning this lignite coal. This means was found by J. Elmer Parkison, inventor of the furnace, and improved upon by Samual G. Porter and William Manhire.

The principle on which the inventors worked was a simple one. Scientists told them that the ordinary lignite coal should give better results than the bituminous coal as there was more gas in it to be consumed. Under the old furnace practice this gas, which is given off at a low temperature, was not consumed in the furnace but went up the chimney in the shape of thick, heavy smoke.

The problem of the inventors was to find some method of burning this gas in the firebox and applying the heat to the making of steam. This was done by a different method of firing the furnace, and instead of a thin layer of coal a deep fire-bed was used. The fire-bed in the Parkison furnace ranged from 18 to 24 in. in depth. This allows for slow combustion of the coal.

In addition to this deep fire-bed the Parkison furnace is constructed in a different manner from the ordinary furnace, in that it has two fire-boxes; the primary fire-box, where the heat from the carbon in the coal is used, and the secondary fire-box, where the gases, that ordinarily are allowed to escape up the chimney, are consumed. The only matter that escapes from the smokestack is the unburnable gas, which is practically the same color as the air, so that the furnace not only consumes everything burnable in the coal, but also acts as the most perfect smoke-consumer on the market.

The ashpit on the Parkison furnace is sealed and the door of it is not allowed to stand open. Only enough air to assist in the actual combustion of the coal is allowed to pass into the primary fire-box. This current is automatically regulated by a fan. With this air, which is introduced under the fire-bed, is a current of exhaust steam. The purpose of this steam is to decompose the coal, with the attending result that there are no clinkers to be removed from the fire-box of the furnace.

Another current of air, also automatically regulated, is allowed to pass into the secondary fire-box to assist in the combustion of the volatile gases that are given off from the coal in the process of burning. The temperature of this fire-box is ordinarily from 300 to 700 deg. higher than in the primary fire-box, showing that the best part of the coal is wasted in the ordinary flat-grate furnace practice.

The Western furnace company was organized to exploit the invention of Messrs. Parkison, Porter and Manhire. The company installed the furnace on a simple guarantee of delivering power from the use of lignite coal. Furnaces were installed under boilers in the Cripple Creek district and under roasters at the mills in Colorado City. Only lignite coal was used at this time and the furnace was exploited purely on its merits as a means to burn lignite coal. A strike in the lignite coal fields of Colorado cut off this supply and only bituminous coal was available. This fuel was used, and the furnace was discovered to be adaptable to it. Further experiments with the different fuels showed that the Parkison furnace could burn the different coals in Colorado at a great saving over the ordinary flat-grate practice or over any automatic-stoker practice in use.

The officers and directors of the Western furnace company then made experiments with the coals from other states and it was found that the furnace would burn at a saving all lignite coal, all the semi-bituminous coals and all the bituminous coals. In other words, it was discovered that the Parkison furnace could burn 90 per cent of the varieties of coal at a guaranteed saving of at least 15 per cent over almost every practice in use.

The International Engineering Corporation was then formed and the Western Furnace Company was made one of its subsidiary companies. The International Engineering Corporation is the parent company of the different subsidiary concerns that are being organized in the different manufacturing centers of America for the exploitation of the Parkison furnace.

The Parkison furnace is adaptable for use under all types of boilers. In addition to the actual fuel saving effected in burning all burnable qualities in the coal, builder's rating of the boiler is guaranteed. In fact, many installations of the furnace have been made on the contract that an overload of the builder's rating and a certain percentage of efficiency should be made in addition to a percentage of saving in the fuel bills.

The method of firing the Parkison furnace, and its different arrangement, not only eliminates clinkers in any kind of coal used, but reduces the percentage of combustible matter in the ashes. The ash removed from the ashpit of the furnace has the appearance of wood ashes, and there is not as much of it as under any other furnace now in use. In addition to this, chemical tests show that the ash will average less than 5 per cent combustible matter.

The Parkison furnace was recently installed in the smelters of the Pittsburg Zinc Company, of Pittsburg, Kan., to take the place of natural gas, which had been used by that concern for many years. The natural gas supply is becoming exhausted and the fact that the company could not renew its contracts for gas at 3 cents per 1000 cu. ft., or, in fact, could not get a guaranteed supply of the gas at any rate, caused the officers of the zinc company to look for a substitute for gas. They investigated the claims made for the Parkison furnace and decided to install four furnaces under one of their zinc roasters.

Using natural gas it cost the Pittsburg Zinc Company $1.75 per ton to roast their ore. This expense was divided as follows:

In a twenty-four-hour day 1,500,000 ft. of natural gas at 3 cents per 1000 cu. ft. cost $45. This roasted 28 tons of ore. The services of two men at $2 per day brought the expense of roasting the 28 tons to $49—or $1.75 per ton.

The Parkison furnaces installed roasts the zinc ore at a cost of $1 per ton. This expense is divided as follows:

Six men at $2 per day, $12; 8 tons of coal at $1 per ton, $8; ore roasted in one twenty-four-hour day, 20 tons; or a total of $20 for the roasting of 20 tons of ore, which makes the cost per ton $1.

The experience of the engineer of the International Engineering Corporation at the plant of the Western Alkali Manufacturing Company, at Green River, Wyo., shows in a convincing way the fact that all the burnable matter in the coal is consumed and only waste material goes into the ashpit. The furnace was installed under a 282-hp boiler. The ashpit was sealed for forty-eight hours and the boiler was forced to deliver 312 hp. At the end of the forty-eight hours one ordinary wheelbarrow load of ashes was removed from the ashpit.

The management of the Vindicator Gold Mining Company, of Independence, Col., in the Cripple Creek district, says that the company saves approximately $1,000 per month with the use of the Parkison furnace under three of their boilers.

The Golden Cycle Mining Company, of Colorado Springs and Victor, Col., has thirty-six Parkison furnaces in its mill at Colorado City and its mine at Victor. The management of the company admits a saving of $1,300 a month under four boilers at Victor, and says that where the coal bill amounted to $3,800 per month before the installation of the furnace it is now only $2,500. At the mill in Colorado City the company has the Parkison furnace under ten Edwards ore roasters. Here the coal bill has been cut practically in two. The saving effected at the Golden Cycle mill is in the neighborhood of $200,000 a year.

The Crescent Mill & Elevator Company, of Denver, Col., is saved between 20 and 30 per cent in the fuel bill by the use of the Parkison furnace. In addition to this saving the engineers at the mill are able to secure an increase of 25 per cent in the coarse grain production and an increase of 15 per cent in the mill with the same boilers through the increased efficiency given by the furnace.

The Pewabic Consolidated Gold Mines Company, of Central City, Col., had the first repairs to make on a Parkison furnace after using it for three years. In addition to this the management of the company admits a saving in the fuel bills effected by the furnace to be over 35 per cent.

The management of the United Hydro-Electric Company, Georgetown, Col., figured that the furnace paid for itself in five months with the saving effected by it. This company supplies the towns of Georgetown and Idaho Springs, Col., with electricity. The plant is near Georgetown, and after the installation of the Parkison furnace the Western Furnace Company received a letter from the town officials of Georgetown thanking it for the elimination of the smoke nuisance from the electric plant through the use of the furnace.

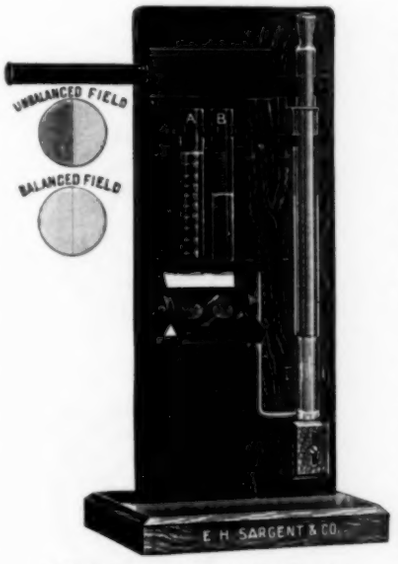
The Bull Hill Mining & Development Company, of Cripple Creek, saves 30 per cent in its fuel bills by the use of the Parkison furnace and, according to the management, has an absolutely smokeless stack.

The Darbyshire-Harvie Iron & Machine Company, of El Paso, Tex., has had the Parkison furnace in its plant for four years. The management of this concern says that the amount expended for repairs during that time has been trifling. According to W. C. Harvie, the manager, a statement that the furnace had saved the company 50 per cent in its fuel bills would be moderate. The furnace burns refuse from the city streets, and runs of ten hours are frequently made using no coal whatever.

The officers of the International Engineering Corporation are Samuel G. Porter, president; H. G. Cunningham, first vice-president; George P. Mason, second vice-president and general manager sales department, and Walter G. Christie, secretary and treasurer.

The Campbell-Hurley Colorimeter

The Campbell-Hurley colorimeter, which is shown in the adjoining illustration and which has just been placed on the market by Messrs. E. H. Sargent & Co., of Chicago, is a modification of the well-known Kennicott-Sargent colorimeter, the modification being in the telescope arrangement by which a person looking through the eyepiece observes a single circular field divided vertically by an almost imperceptible line when the two solutions are of the same intensity. By manipulating the plunger the level of the liquid in tube B can be easily raised or lowered, thus causing the right half of the image to assume a darker or lighter shade at will. The unknown solution is placed in the left-hand tube A, which is 19 centimeters long and 3 centimeters in diameter, and graduated for 15 cm., this graduation being subdivided into 10 divisions. The standard solution is placed in the right-hand tube B, which is the same size as A, the graduated portion being divided into 100 divisions of 1.5 millimeters each. The tube B is permanently connected by a glass tube with the reservoir, in which works the glass plunger. As the tube B and reservoir are made in one piece the liquid used for the standard solution comes in contact with glass only, thus preventing any possibility of chemical change due to contact with the container.



CAMPBELL-HURLEY COLORIMETER.

Tubes A and B, with the connecting reservoir, are held in position by the spring clamps, which allow the glass parts to be readily removed for cleaning and filling. The light for illuminating the solution is reflected upward through tubes A and B by means of the adjustable mirror. The light passing through these tubes impinges on two mirrors cemented to brass plates sliding in grooves cut at an angle of 45 deg. in the sides of the wooden box. This box is supplied with a loose-fitting cover, thus allowing easy access for the purpose of removing and cleaning the mirrors. One-half of the circular field of light from the tube B is cut off by one of the mirrors, the vertical edge of which acts as a dividing line between the two halves of the circular field. The image of the half of the tube B is then observed in juxtaposition to the opposite half of the image of the tube A. Concerning other details reference may be had to the *Journal* of the American Chemical Society, July, 1911.

Hydrocarbons in California.—According to the statistics of the California State Mining Bureau, hydrocarbons—including petroleum, natural gas, bituminous rock, exclusive of asphalt and all other refined products—were produced during 1911 in California to the value of $41,161,226. This includes: petroleum sold in 1911 or in stock at the end of the year $37,920,820, petroleum used as fuel in the field $2,631,268, natural gas $491,859, bituminous rock $117,279.

A New Vacuum Pump.

A new vacuum pump which should be especially valuable for chemical purposes has been placed on the market by the May-Nelson Rotary Pump, 508 Wilkins Building, Washington, D. C. It is of compact construction, not liable to damage, and capable of producing quickly a reasonably good vacuum, say a pressure of 0.1 to 1 mm of mercury or about one-thousandth of an atmosphere.

The May-Nelson rotary pump works on a new principle and differs considerably from any other pump now on the market.

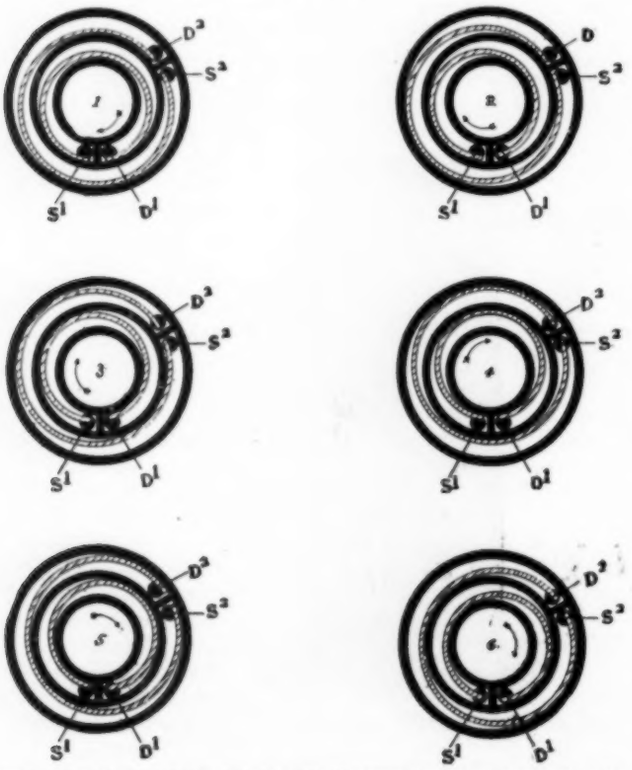


FIG. 1.—DIAGRAM ILLUSTRATING SUCCESSIVE STEPS IN OPERATION OF VACUUM PUMP.

It consists essentially of three parts only—the body, the impeller, or movable part, and the shaft, the whole being so arranged that rotation of the shaft imparts an eccentric motion to the impeller. In both body and impeller are cut a series of accurately concentric, equidistant, and similar grooves (for vacuum work, either two or three), so arranged that each of the projecting rings thus formed on either part dips into a corresponding groove on the other part. In each of the grooves cut in the body of the pump is inserted a radial dividing wall, on either side of which are drilled down the two parts corresponding to that ring; and corresponding portions of the projecting rings are cut away.

The mode of action of the pump depends upon the rolling contact made between the external cylindrical surfaces of either piece and the adjacent internal cylindrical surfaces. The principle is readily seen from Fig. 1, which represents a right section through body and impeller, in a plane close to the foot of the grooves in the body, in a series of six equidistant positions during one complete revolution of the shaft.

In the diagrams the cross-hatched rings represent the projecting rings on the impeller, which dip down into the corresponding grooves cut in the body of the pump, and in action roll against the adjacent cylindrical surfaces, those, namely, of the solid black rings.

In each diagram the position of the shaft is one-sixth of a revolution ahead of that in the preceding diagram. S_1 is the port connected with the vessel to be evacuated; D_1 is connected with S_2, and D_2 is the discharge.

Suppose that the direction of rotation of the shaft is such that the rolling contacts progress in the clockwise direction, and consider, for the time being, only the inner impeller ring and the contact which it makes with the innermost black ring. From the series of diagrams it is evident that the advance of this rolling contact is accompanied by the formation, adjacent to the port S_1, of a crescent-shaped space; this space first increases in size, thus sucking air in through the port S_1, is then successively closed off from S_1 and brought into connection with the discharge port D_1; finally, it decreases in size, thus forcing the air out through D_1. At the same time, though half a revolution later, the same process is going on at the contact between the outer cylindrical surface of the inner impeller ring and the surface adjacent to it (the inner surface of the middle black ring). Each ring is therefore double-acting; that is, it is effective at both its inner and outer cylindrical surfaces.

The air forced out through D_1 passes to S_2, thence round the outer ring, and is ultimately discharged through D_2. We thus have in such a pump two double-acting rings working in series. The dividing walls are set 120 deg. apart, in order that several of the contacts may always be effective obstacles in preventing the air from rushing backward; in other words, this arrangement prevents back-firing. The action of the three-ring pump is the same; the air passes round a third ring in series with the other two before it is finally discharged.

The action of the May-Nelson vacuum pump, therefore, depends upon the fact that the rolling contacts between the adjacent cylindrical surfaces push out the air ahead of them and ultimately force it out of the discharge ports. That the arrangement made use of in this pump should suck air at all seems at first sight somewhat surprising. Its effectiveness would appear to be due to the fact that we have a rather long contact, since it is between two surfaces of nearly the same radius of curvature; this, combined with the considerable viscosity of air and the high speed at which the pump may be driven, insures that the air is always pushed ahead of the rolling contact, and thus determines the action of the pump.

Fig 2. is a vertical section through the pump, scale about one-third. Driven by a ¼-hp., 800-r.p.m. motor, this pump evacuates a vessel holding 6 liters down to a pressure of about 0.1 mm of mercury in less than 2 minutes. It is capable of reducing the pressure, as measured on the McLeod gauge, down to 0.5 mm of mercury. Under favorable conditions it will

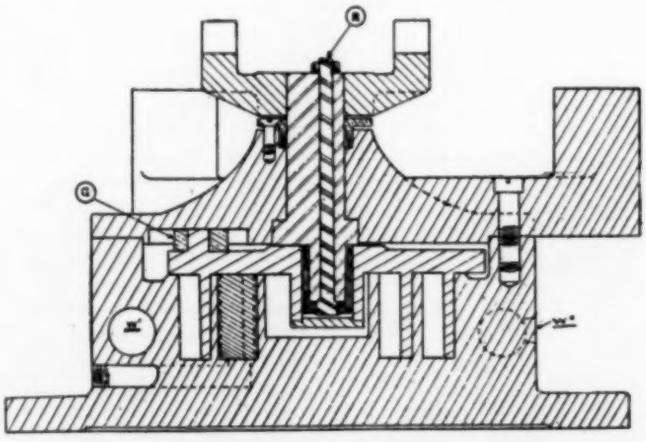


FIG. 2.—SECTIONAL DIAGRAM OF VACUUM PUMP.

produce pressures of the order of one-hundredth of a millimeter. Connections for efficient water cooling are provided; the cooling water circulates in the core W_1W_2, but it need not be used unless the pump is to be run for several hours continuously.

The section of Fig. 2 which is in the plane of the dividing wall on the inner ring exhibits the general dimensions of the rings and grooves and illustrates the mode of adjustment. When adjustment is required (which will be very seldom) it is necessary only to unscrew the lock nut on the eccentric adjustment rod R, to loosen this adjustment by tapping with a light hammer, and then to rotate the rod R clockwise until one feels that the impeller just touches the body of the pump. This figure also shows the guide-blocks G, the function of which is to prevent the ends of the impeller rings from knocking against the dividing walls on the pump body.

Fig. 3 is a view of the pump, complete with motor (scale about one-sixth). It shows the pump ready for action. A yoke joint is employed for connecting the pump with the motor; its employment saves trouble in accurately centering the motor, and admits of a slight vertical motion of the impeller.

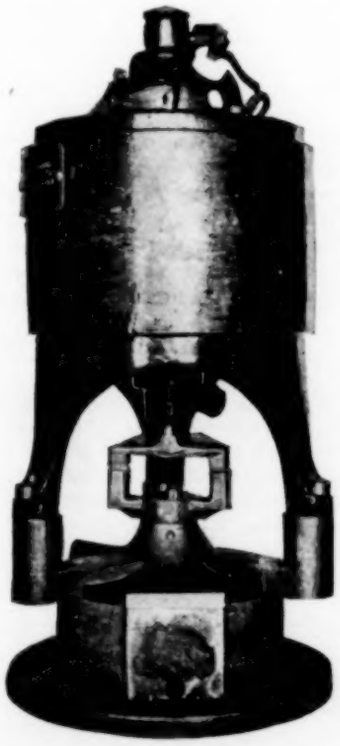


FIG. 3.—OUTSIDE VIEW OF VACUUM PUMP AND MOTOR.

The vessel to be evacuated is connected to the threaded hole.

The mechanical advantages of the pump are due to its compact construction. It has only one moving part, besides the driving shaft; hence it is easy to work and always in order. It is unbreakable, is direct connected to the motor, and requires no especial base; it may be set anywhere and does not need to be fastened down. It is entirely valveless, and in consequence requires but very little attention, lubrication or adjustment. Its action is rapid, and yet it gives a vacuum sufficient for all chemical, and for most other, purposes.

By leaving the suction side open, the pump produces a blast sufficient for several laboratory blowpipes; when driven by a motor of sufficient power the pump can be used to produce air pressure up to 100 lb. per square inch.

For chemical purposes the following fact is of especial interest: Water, or, indeed, any liquid which does not corrode the metal (iron), may be sucked through the pump without ill effect; in a very few minutes the pump will eject it all and evacuate as satisfactory as ever. This behavior renders this pump especially suited for vacuum distillations, since no intermediate condenser is required.

Electric Transformer Crucible Furnace.

To supply the demand which exists for a furnace midway between the small laboratory electric furnaces with a capacity measured in grams and the large commercial electric furnaces, the Allgemeine Elektricitäts Gessellschaft of Berlin has developed a furnace type which has just been placed on the market under the name of "transformer crucible furnace." With the exception of the smallest type, these furnaces are constructed in sizes for taking charges from 1 kg (2.2 lb.) up to 100 kg (220 lb.).

Each furnace unit is a combination of a transformer with a carbon crucible. The crucible itself forms the resistor. The secondary terminals of the transformer are connected by flexible copper conductors with two water-cooled contacts gripping the crucible, so that the secondary current from the transformer passes through the crucible and heats it. The carbon crucible is lined in a special way to prevent the carbon from having any effect on the charge and to prevent the current from passing through the charge.

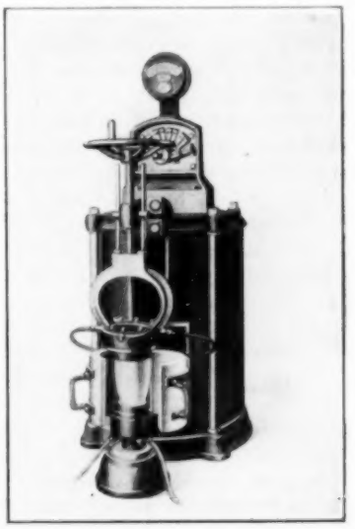


FIG. 1.—LABORATORY ELECTRIC CRUCIBLE FURNACE.

The transformer can be regulated so that up to twenty-four different rates of heating can be obtained. Besides the regulating switch an ammeter is provided for all except the smallest sizes.

This furnace has already been introduced in Germany into various industries, especially in gunmetal, copper, bronze, and iron foundries, for melting precious metals, glass, and enamels, in steel laboratories, and chemical and other industrial establishments.

The smallest size of this furnace, type TO, 00 has been designed specially for the use of dentists; if required tappings can be provided which enable the furnace to be operated at 1000 deg., 1400 deg., and 1800 deg. It is used especially by dentists for melting enamel, porcelain, gold, platinum, and the compounds employed for artificial teeth. All expensive auxiliary apparatus, such as ammeters, etc., are dispensed with in this type. The only part subjected to wear, is the carbon crucible, which is cheap and can be easily replaced by another one.

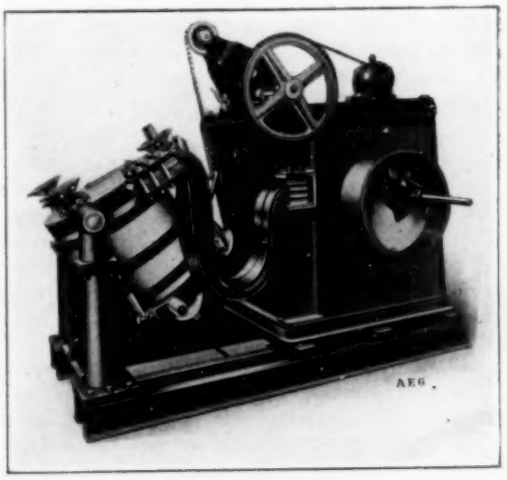


FIG. 2.—ELECTRIC CRUCIBLE FURNACE WITH TILTING DEVICE.

Fertilizer Convention.

The National Fertilizer Association and the Southern Fertilizer Association held their 19th annual convention in Atlantic City from July 15th to 19th.

The program of addresses and papers included the presidential address by President W. W. Hubbard, of the Peerless Fertilizer Company, of Chestertown, Md.; an address by Dr. Frank K. Cameron, of the U. S. Department of Agriculture.

on the present status of fertilizer enquiries, practical and theoretical; by Mr. W. H. Waggaman, of the Bureau of Soils, on the new phosphate deposits of America; by Prof. H. G. Bell, on the work of the Middle West Soil Improvement Committee of the National Fertilizer Association, and by Mr. Hoyt S. Gale on the U. S. Geological Survey's searches for possible sources of potash. The program also included two papers by Mr. J. S. Brogdon, Atlanta, Ga., on "the chemist versus the analyst," and by Mr. E. P. Verner, of Charleston, S. C., on "organic ammoniates and their availability," as well as addresses by Mr. W. W. Finley, president of the Southern Railway; Congressman A. F. Lever, and Prof. Andrew M. Soule, of the Georgia State College of Agriculture.

The entertainment committee went to a great deal of trouble to give the members and guests a good time, and was very successful in accomplishing this result. Young's Million Dollar Pier was the scene of most of the activities. A very interesting and carefully prepared Fertilizer Exposition on this Pier proved a particularly pleasant feature of the convention.

At the entrance of the exhibition was the fine exhibit of the Sturtevant Mill Company, which had the largest display of all. Other attractive and effective exhibits were made by Walker & Elliott, Stedman's Foundry & Machine Works, Kent Mill Company, Automatic Weighing Machine Company, H. W. Johns-Manville Company, American Cyanamid Company, Gardner Crusher Company, Gruendler Machine Company, Main Belting Company, Duryea Manufacturing Company, Jeffrey Manufacturing Company, Lehigh Car Wheel & Axle Company, Wiley & Company, Norton & Company, Schutte & Koerting, and many others.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Ore Treatment (Continued).

568,099, Sept. 22, 1896, Louis Pelatan, of Paris, France, and Fabrizio Clerici, of Milan, Italy.

Relates to an apparatus for the extraction of gold or silver from their ores or compounds. The ores are pulverized and mixed with a solvent such as alkaline cyanid containing some sodium chlorid. The electrolytic cell consists of a rectangular vessel in which is a traveling endless belt, the surface of the belt provided with agitating arms. The belt may be made of metal and connected as an anode, or the belt may be of non-conducting material and a metallic anode inserted in the cell. The cathode is of mercury or of amalgamated metal, such as copper. In the operation, the endless belt, with its agitating arms, keep the ore and solution in constant agitation and the electric current assists in extracting the precious metals. When the operation is completed, the gold may be extracted from the mercury in any usual manner.

568,724, Oct. 6, 1896, Emile Andreoli, of London, England.

Relates to an electrolytic cell for decomposing cyanide of gold solution or similar solutions. The cell may be divided into compartments by suitable diaphragms and the bottom of the cell covered with mercury connected as cathode. A plurality of perforated plates or wire gauze sheets are electrically connected to the cathode. The cyanid of gold solution is applied to the cathode compartment and in the positive compartment there may be cyanid of gold solution or preferably a caustic potash or caustic soda solution. The anodes may be of iron or peroxidized lead, such as a positive plate. The cathodes are amalgamated by contact with the mercury, and present an enormous surface to the cyanid of gold solution. The gold solution enters the tank at the bottom above the mercury cathode and escapes in outlets at the upper part of the cell, from which is supplied to a second cell where the electrolysis is repeated. When the operation is completed, the plates are cleaned up and mercury recovered and retorted in the ordinary way.

568,741, Oct. 6, 1896, Henry R. Cassel, of New York, N. Y. York.

Relates to a process for extracting gold from its ores and other compounds by means of bromin solutions and means to regenerate such solution. A solution of a bromid of a suitable alkali metal is electrolyzed in a decomposing-tank containing anodes and cathodes of carbon or any material not affected by the solution. The cell also contains a diaphragm of asbestos forming anode and cathode compartments. The electrolysis accumulates bromin in the anode compartments, while potassium hydrate accumulates in the cathode compartment. Water is also decomposed, oxygen being set free at the anodes and there forms, with the bromin, an oxygenated or oxybromid solution. The pulverized ore, with the bromin solution, is placed in a rotating cylinder and therein thoroughly mixed until all the gold has been dissolved. After filtering, the tailings are leached by means of the potash solution above referred to. The potash combines with the bromin and with the bromid of gold remaining therein, and both are leached out The gold bromid solution is run through scrap iron and pieces of carbon and the gold rapidly precipitated thereby in a finely divided state. The resulting bromid solution from the iron and carbon treatment is re-electrolyzed, the process being continuous.

568,843, Oct. 6, 1896, Victor Engelhardt & Adolf Nettel, of Vienna, Austria-Hungary.

Relates to the extraction of metals, such as antimony, arsenic, and mercury, from their sulfur compounds. The metallic sulfid or compound is treated in a tank provided with a stirrer, with a soluble sulfid of the alkali or alkaline earth metals in aqueous solution, in order to convert the metal into a soluble double sulfid. The resulting solution of double sulfids is filtered, and the solution then decomposed by electrolysis, the metal being obtained in a solid form and sulfureted-hydrogen gas liberated.

571,468, November 17, 1896, Thomas P. Barbour, of San Antonio, Texas.

Relates to processes of treating gold and silver ores and recovering their metals by chlorination. The ore is first pulverized to eighty-mesh fineness and is then fed into a chlorinating tank. Within the tank, at the ends thereof, are electrodes of copper serving as poles for the transmission of the electric currents through the fluid pulp. The tank charged with the proper quantity of pulverized ore has added to it finely pulverized black oxid of copper in the proportion of about six or eight pounds to a ton of the ore and mixed with sufficient water to make a pulp of the consistency of thin mortar. To this is added sulfuric acid in a quantity equaling about two-thirds of the weight of the copper oxid added. The tank is then rotated for about an hour and a half. After this treatment, the rotation is stopped and a second charge introduced into the same, consisting of eight pounds of black oxid of manganese, thirty pounds of salt and twelve pounds of sulfuric acid. The tank is again rotated and an electric current is passed through the pulp until the chlorination is complete, after which the circuit is broken and the tank completely emptied. The gold and silver of the ore are held in solution as chlorids and are removed from the chlorinating vessel. The remaining pulp and wash-water are conveyed to the metal-precipitating tank, heated by steam which contains agitating-arms of metallic zinc. The electric current is passed through the solution from the agitating arms to electrodes at the periphery of the tank. The electric current quickly precipitates any remaining metallic silver and the zinc also serves to precipitate silver. When the precipitation is complete, the current is discontinued and the solution drawn off. The tank is now filled with water and agitation again commenced and continued for a suitable time, after which the pulp is allowed to

settle and the clear solution drawn off into a vessel containing the original gold solution. More water is now added to the pulp to give the same a suitable consistency and salt and granulated zinc are added. The mass is now taken to an amalgamating-pan for treatment. Mercury is added in the usual way, and the agitation in the amalgamation-pan is kept up until treatment is complete. The amalgam is collected and retorted as usual. To the gold solution, above noted, is added ferrous-sulfate solution, and the gold precipitated as a dark brown powder which is collected and fused. If copper is present in the ores, the solution from the zinc treatment is now treated with scrap-iron in the ordinary manner for precipitating copper.

573,233, Dec. 15, 1896, Max Netto, of Almazarron, Spain.

Relates to processes for precipitating silver and gold from their cyanid solutions and regenerating the cyanid solution. A cyanid solution containing gold and silver in the form of alkali double cyanids is acidulated with hydrochloric acid until a slightly acid reaction is obtained. Silver chlorid is thereby precipitated while the gold remains in the solution. The precipitation of silver chlorid is obtained according to the following equation, in which, for example, the combination of potassium cyanid with silver cyanid is assumed:

$$AgK(CN)_2 + 2HCl = 2HCN + KCl + AgCl.$$

The solution from which the silver chlorid has been separated still contains gold and is subjected to electrolysis, whereby this metal is deposited on the cathodes. Care has to be taken that the gold-containing solution contains a sufficient quantity of hydrochloric acid. After electrolysis, the solution is next mixed with caustic alkali until a slightly alkaline reaction takes place and the alkali cyanid thereby regenerated.

NOTES.

Ozocerite deposits occur in Utah in the vicinity of Colton and Midway Station, near Soldier Summit. The manner of separating the mineral from the associated shale and sandstone is to crush the rock and subject it to heat in boiling water. Ozocerite melts at from 54 to 70 deg. C., rises to the surface of the water and flows into receiving vessels. After cooling it is again melted to remove moisture, and is then cast in molds.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in April, 1912, was sixty-one. The quantity of ore milled during that period was 2,106,150 tons. There were 9991 stamps in operation, with an average duty of 8.20 per twenty-four hours. Tube mills in commission numbered 263. The yield for the month was 737,660 fine ounces gold. The quantity of ore treated and the gold recovered show a slight decrease compared with the preceding month, but a large increase over the corresponding month of last year.

A map of the Western mining districts has been prepared by the United States Geological Survey. Many of these mining districts are well known and have well-recognized boundaries, but others are more obscure. Bulletin 507, of which the map is a part, contains an introduction on geology and two plates showing the area covered by the Survey in geologic and topographic work. A separate map of each of the fourteen states shows the location and area of over 1400 mining districts. California is shown in two maps, the northern portion containing the largest number of mining sections, viz., 222. The smallest is the map showing the trans-Pecos section of Texas and contains only eleven districts. Copies of the Bulletin can be obtained free on application to the Directory of the Survey, Washington, D. C.

Tin and aluminium hold different positions in popular estimation. The former is regarded as quite common and the latter as less so. The price of the former is, however, twice that of the latter. The annual production of aluminium is far below that of tin, but the rate of production of the former is increasing about fifteen times as fast as the production of the latter. Twenty years ago aluminium cost about $1 per pound in Europe, having then just dropped from $4. At the same time tin cost 18 cents per pound. To-day the position is reversed, and tin is now worth in this country about 45 cents and aluminium 23 cents per pound.

Venezuelan iron ore mined by the Imataca Ore Company is said to be of a superior grade, averaging as high as 67 per cent iron.

The tenor of zinc and lead sulphide concentrates produced in Missouri in 1911 is shown in the following tabulation, which has been prepared by the United States Geological Survey:

	Per Cent Crude Lead.	Metal Ore, Zinc.	Per Cent Lead in Lead Conc.	Per Cent Zinc in Zinc Conc.	Value per Ton. Lead Conc.	Value per Ton. Zinc. Conc.
Southwest:						
Soft ground..	0.3	2.3	79.9	58.9	$54.41	$40.80
Sheet ground.	0.4	1.24	79.8	58.6	56.50	39.79
Southeast	3.7	....	67.3	....	46.94	.....

The Bureau of Mines has recently published some interesting pamphlets. "Government Coal Purchases Under Specifications" is the title of Bulletin 41. "The Slagging Type of Gas Producer" is discussed in Technical Paper 20. "The Behavior of Nitroglycerin When Heated" is the subject of an investigation published in Technical Paper 12.

The First National Copper Company, which owns a copper smelter in California, is reported to be contemplating measures for re-opening the plant, which was closed in the recent agitation against smelter smoke. If possible, steps will be taken to bring the operation within the limit of the law.

The Ford Motor Company, of Detroit, Mich., for their new foundry for making automobile engines have contracted with the Rockwell Furnace Company, New York, for one of the largest installations of small core ovens in the automobile field. This installation consists of 24 ovens containing 88 shelves. The ovens are of the type known as the "reel type" core ovens, each shelf being a complete circle about 5 ft. in diameter, turning independently on a central axis. One-half the shelf is in the oven with the cores on it being baked while the other half of the shelf is outside, serving as a rack. Producer gas will be used as fuel. This type of core oven is especially adapted for automobile work in that it does away with the jar and jolt which arises when car-type ovens are used.

Water-proof Paints.—The 1912 edition of the "Red Book" of Messrs. Toch Brothers of New York City has just been issued. Profusely and nicely illustrated, it tells an interesting story on the Toch R. I. W. paints in an interesting manner. R. I. W. stands for "remember it's waterproof." These paints are the outcome of 64 years of researches and experience of Toch Brothers. Details are given of the properties of various of these paints for damp-proofing, water-proofing, steel protection and decorative purposes, with illustrations of some of the many modern buildings and skyscrapers in which the R. I. W. paints are used for such purposes.

PERSONAL.

Mr. W. J. Sharwood, chemist for the Homestake Mining Company, Lead, S. D., has been on leave of absence for six months, during which time he has been making investigations for the Bureau of Mines. He was in Denver about the middle of July en route west.

Mr. Walter L. Reid, superintendent of cyanide plant for the Smuggler-Union Mining Company, Telluride, Col., recently made a trip to Oregon, where he is erecting a modern cyanide mill.

Prof. S. B. Christy, dean of the school of mining, University of California, has been re-elected corresponding member of the Institution of Mining and Metallurgy, London.

Mr. D. N. Muir, who has charge of the interests of the United States Smelting, Refining & Mining Company in Arizona and at Needles, Cal., has removed his office from Kingman to Gold Rock, Ariz.

Mr. Arthur H. Carpenter, of Denver, has accepted a position on the chemical staff of the American Vanadium Company, Pittsburgh, Pa.

Mr. M. W. von Bernewitz, metallurgist of Kalgoorlie, Western Australia, sailed early in July for San Francisco.

Dr. F. G. Cottrell, of San Francisco, who is in charge of metallurgical investigations of the Bureau of Mines, expects to attend the International Congress of Applied Chemistry in New York in September. His route east will take him through Montana and Michigan.

Mr. D. C. Jacking, of Salt Lake City, has gone to Alaska. Before starting north he made an inspection of the Butte & Superior Company's mine and mill at Butte.

Mr. Arthur H. Roller, of Idaho Springs, Col., manager for the Hudson Reduction Company, has been appointed to a similar position with the Argo Ore Purchasing & Reduction Company and will build a new mill for the latter company.

Mr. Victor G. Hills, of Denver, is making a professional trip into the Yukon Territory, Canada.

Mr. E. C. Van Diest, of Colorado Springs, Col., has been appointed by the governor of the state to serve as trustee of the Colorado School of Mines, vice Franklin Guiterman, resigned.

Mr. W. W. Fisher, chief engineer of the Standard Underground Cable Company at Perth Amboy, N. J., was elected president at the meeting of the fifth international convention of the Esperanto Association of North America.

BOOK REVIEWS.

Systematic Treatment of Metalliferous Waste. By **L. Parry.** 12 mo., 14 x 21½ cm., 121 pages; price 5 shillings (in New York, $2.00). London: The Mining Journal.

An interesting book for the metal handler, poorly written as far as style goes, somewhat alchemistical in its formulae and terminology, rambling in plan and unsystematic in detail, and yet worth while. Without knowing the author, the book seems to warrant the guess that he is an eminently practical man of the old school, one who has known and worked with metals, drosses, wastes and junk all his life, who has not been blessed with a modern chemical education, but who knows metals by that long experience which engenders intuitions.

The materials considered are lead, tin, antimony, copper, nickel, and zinc wastes of all sorts. The information about reducing and refining these is very practical, but the reader must dig it out, avoiding being misled by odd forms of expression and queer ways of putting things. The author's experience and enthusiasm atone for the many literary defects of the book, and his willingness to impart his hard-won knowledge and to share it with the world are certainly worthy of commendation.

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Die elektrolytische Alkalichloridzerlegung mit starren Metallkathoden. By Dr. **Jean Billiter.** Part I: Patent Literature. 16 x 24 cm., 284 pages, 189 illustrations; price, 16.50 marks (in New York, $5.50). Halle: Wilhelm Knapp.

This is a compilation of German, British, and American patents on the electrolytic decomposition of alkaline chlorides, using solid metal cathodes. In each case the patent claims are given *verbatim,* followed in most cases by an illustrated description of the subject matter of the patent. Part II., which is to appear this year, will contain a description of industrial plants operating these processes.

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Eisenhochofenschlacken, ihre Eigenschaften und ihre Verwendung. By Ing. Chem. **Hans Fleissner.** 19 x 28 cm., 57 pages, 32 illustrations, 3 plates; price, 3 marks (in New York, $0.75). Halle: Wilhelm Knapp.

This brochure is directed particularly towards the qualities of blast-furnace slag, which determine its suitableness for Portland cement. The estimation or description of slag from this point of view is very practical whenever cement is to be made from them, and it also gives a new point of view from which to regard slag characteristics. For the latter reason the wide-awake blast-furnace metallurgist will find his knowledge of slags in general increased by reading this book.

The labors of Vogt and Doelter are satisfactorily utilized in explaining the constitution, fusibility diagrams and surfaces are discussed, and texture and color are considered.

The practical part includes granulation, utilization for slag-wool, pumice, glass, road-metal, bricks, hydraulic cement, and Portland cement. The information contained is valuable to any man interested in these lines.

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Der Praktische Hochofenbetrieb. By **Theodor Bender.** 17 x 24 cm., 124 pages, 16 illustrations; price, 2 marks; in New York, $2.10. Halle: Wilhelm Knapp.

An eminently practical book on the blast furnace by a practical metallurgist. The various chapters deal with the building of the furnace and accessories, raw materials, products, blowing-in, regular running and changing the running, repairs, interruptions, banking, blowing-out. Throughout economic considerations are more in evidence than scientific explanations. The spiegeleisen practice, discussed at some length, will particularly interest our blast-furnace men. Our chief criticism is that the operation discussed is rather small-scale practice in Germany, which does not fit exactly the general economic and commercial conditions of America.

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Gemeinfassliche Darstellung des Eisenhüttenwesens. By the Verein Deutscher Eisenhüttenleute, 8th edition. 15 x 24 cm., 416 pages, 63 illustrations, numerous diagrams; price, 5 marks (in New York, $1.60). Düsseldorf: Verlag Stahleisen, m.b.H.

The first part, 180 pages, is beautifully printed and illustrated on calendered paper, and forms an elementary treatise on the metallurgy of iron. It is well written and the illustrations are works of art. This part alone is well worth the price of the whole book to a student or general reader.

The second part, over 200 pages, contains commercial statistics of the iron industry, including coal and coke, in 30 different countries, brought up to the close of 1910; also statistics of international commerce in ore, iron, steel, and scrap, with information as to tariffs brought up to 1912.

An appendix of 30 pages gives more detailed statistics of iron and steel works and foundries in Germany. It is a most excellent work, cheap and good.

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Chemical Works, Their Design, Erection, and Equipment. By **S. S. Dyson** and **S. S. Clarkson.** Octavo, 15 x 25 cm., 207 pages, 80 illustrations, 9 plates; price, 21 shillings (in New York, $7.50 net). London: Scott, Greenwood and Son. New York: D. Van Nostrand Company.

The authors have aimed to give the fruit of twenty years' experience in designing chemical plants, including even working drawings of typical installations. Starting with the site, foundations, roads and trackage, the general lay-out of the plant is discussed, then power house and other accessories.

The actual chemical plant is illustrated by the following types: Sulphuric acid, hydrochloric acid, nitric acid, high explosives, ammonium sulphate, fertilizer works. The concluding chapters deal with other generally used equipment not previously described, saw-mill and box department, the Alkali and Chemical Works Regulation Act of Great Britain, and general remarks on profit sharing.

It is an exceedingly practical book, which will be highly valuable to the beginner in chemical engineering and useful to every works manager or engineer in the chemical engineering profession.